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PROFESSOR BRUCE CHALMERS, *Harvard University*

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A FIRM STEP TOWARD COMMUNISM

Many years will pass but the events of 1959, the first year of the Seven-year Plan, will not be blotted from the memory of our country and all humanity. The historic Twenty-first Congress of the Communist Party of the USSR, which took place at the beginning of the year, developed and outlined a program of a large-scale building of a communistic society in our country.

The firing of three cosmic rockets, the launching of the first atomic ice-breatker in the world, the "Lenin", have all indicated what unprecedented successes Soviet scientists, engineers,and workers have attained.

The historic visit of the head of the Soviet government, Comrade N. S. Khrushchev, to theUnited States of America, justly called a mission of peace , has aroused great hopes in the hearts of millions of people. The Soviet people and all progressive humanity are deeply grateful to Nikita Sergeevich Khrushchev for his selfless labor in the name of peace on earth.

An important event of 1959 has been the third session of the Supreme Council of the USSR, which approved a plan for the growth of the national economy and the state budget of the country for 1960. The papers of the session and speeches of the deputies clearly demonstrate with what enthusiasm the Soviet nation is bringing to life the historic resolutions of the Twenty-first Congress of the Party and the June Plenum of the Communist Party of the USSR.

According to the preliminary data the rates of growth of industrial production in 1959 considerably surpass those contemplated for the seven-year annual average quota. A growth of industrial production of 11-12% is expected, instead of 7.7% according to the plan. The capital investment this year exceeds the level of last year by about 25 billion rubles. Heavy industry, the basis of our economy, is growing at high rates. The production of means of production is increasing in 1959 by 12%, that of consumer goods by 10.5%. The production of cast iron and pipe is increasing considerably.

The success obtained in fulfilling the plan for the first year of the seven-year period, and the additional reserves of productive growth revealed in the course of competition,have enabled us to set greater goals in the plan for 1960 than were comtemplated in the calculations for the seven-year period.

During the first two years of the seven-year period, the plan for industrial production will be over-fulfilled by about 100 billion rubles. For 1960, it is proposed that almost three million tons of steel be smelted, and that almost two million tons of rolled iron, more than was taken into consideration in calculating the preliminary figures of the Seven-year Plan, be produced.

Thus, the rate of growth exceeds the figures set by the Twenty-first Congress of the Communitst Party of the Soviet Union.

The state plan for the growth of the national economy for 1960 places large and responsible tasks on metallurgists. During next year a large step forward should be made on the road to complete satisfaction of the national economy's requirements in metal.

The plan for 1960 provides for increase in the production of cast iron by 9%, steel by 8%,rolled iron by 7% and pipe by 12%. Metallurgists should smelt almost 15 million tons of steel, substantially increase the output of alloy and low-alloy steels, and raise the output of the economic and relief profiles of rolled iron.

In attributing vast importance to the further increase in the yield of petroleum and gas the Central Committee of the Communist Party of the Soviet Union made a special decision regarding increase in the production of pipe during the seven-year period by more than two times.

It is planned that in 1960 important measures in introducing new techniques and advanced technology be carried out. The first 600-ton automated open-hearth furnaces will start up, as will very large-scale blast furnaces, among them a blast furnace with a useful volume of 2000 m^3; there will be considerable expansion in the use of natural gas and oxygen blasting in the production of cast iron and steel. Further work will be done on mastering continuous pouring of steel. In particular, it is proposed to introduce in the Don Basin the largest equipment in the world for continuous steel pouring.

Metallurgists have greeted with great enthusiasm the decisions of the third session of the Supreme Council of the USSR. Workers, engineering technicians,and employees of ferrous metal enterprises warmly approve the plans for the growth of the national economy and the State Budget of the USSR for 1960. In numerous meetings in the factories, teams, and sections, they are introducing valuable suggestions and expressing critical observations directed toward further improvement of the entire productive and economic activity.

With each day, competition is spreading among the foundry, steel and rolling-mill workers-an entire army of Soviet metallurgists-for fulfillment of the 1959 plan ahead of time. In fact, the patriotic movement of shock

workers and teams of Communist Labor has attained vast importance in ferrous metal enterprises.

Thus, in the steel works and mining enterprises of the Kemerovo district, struggling for the right to be called Teams of Communist Labor are more than 40,000 men, in the Sverdkivsk district about 20,000, in the Chelyabinsk and Dnepropetrovsk more than 17,000 each, in the Stalinsk 12,000, in the Kuznetz metallurgical combine more than 4,000, and in the Dzerzhinskii factory 1800 men.

Teams of one of the open-hearth furnaces of the Nizhne-Tagil' metallurgical combine, headed by steel founders Ya. Kal'nichenko, Yu. Ploskonenko, Yu. Zashlyanin, and T. Obraztsov, have pledged themselves to attain in 1963 the level of steel production planned for 1965 and to achieve in 1963 12 tons of steel per square meter of hearth, compared with 10.59 tons in 1958. To accomplish this, complex and individual plans have been made for increasing the pouring of steel. Among the measures contemplated, are the utilization of automatic control of the thermal regime, the supplying of oxygen through facing structures directly into the bath, shortening the duration and increasing the weight of the melt, reducing idle time, reducing the consumption of fuel and increasing the stability of the crowns of furnaces introduced in 1959-60 by hopper charging of free-flowing materials, the mechanization of steam-blast cleaning, etc.

The furnace collective, already, in the first half of 1959, produced more than 2000 tons of steel above the quota.

At the Dzerzhinskii steel works, the teams of senior blast-furnaceman P. P. Lygun assumed the obligation of producing above the quota, in 1959, 2000 tons of cast iron, of saving 8,000 tons of coke, 2000 tons of limestone, 1000 tons of ores and agglomerate, and of lowering the cost of each ton of cast iron by two rubles. During the first six months of 1959, the team produced 1996 tons of cast iron over quota, attained a k.i.p.o.* of 0.658 at a plan of 0.705, saved 6387 tons of coke, 4156 tons of limestone, and lowered the cost of a ton of cast iron by 15 rubles, 50 kopecks. All members of the team pledged themselves to serve as models in production, and to systematically elevate their ideological, political, educational, and technical ways of life.

At the Enakiev metallurgical factory, great success was attained in the competition for the right to be called a Collective of Communist Labor by the team of M. M. Ivanov, foreman of rolling-mill 280. At the Zaporozhe metallurgical factory "Zaporozhstal", no less success was attained by the team of steel founder Ivan Kaela.

Collections of 79 teams, sections, and shifts, of the Zaporozhe metallurgical factories have, by strenous effort, won the right to raise at their places of work the scarlet pennant and hang up a placard with the proud legend, "Here works a Collective of Communist Labor".

Tens of collectives have earned this right in the metallurgical factories of the Sverdlovsk, Stalinsk, Chelyabinsk, Dnepropetrovsk, Kemerovo and other districts.

The new serious tasks presented to metallurgists in the plan for development of the national economy in 1960 require every possible improvement in the organizational work of factory and mining committees of the trade union, directed toward further development of creative initiative and still greater drawing of workers into competition to fulfill the Seven-year Plan ahead of time. It is necessary to expand the struggle for the right to be called a shock-worker and a Collective of Communist Labor, and to make this patriotic movement a massive one. It is necessary to striveso that the great majority of workers can actively participate in the competition for designation as teams and shock workers of Communitst Labor, in order that their productive attainments and high moral qualities may serve as an example and call forth a striving in the broad masses to work and live in the Communist manner.

Not long ago the Presidium of the Central Committee of the trade union of workers in the metallurgical industry held group conferences on the exchange of experience in the organization of socialist competition for the title of teams and shock-workers of Communist Labor and the management of this movement on the part of the trade union committees.

In its resolutions regarding the results of these conferences the Presidium of the Central Committee of the trade union set the organizational work on further development of competition for the title of teams and shock-workers of Communist Labor as the most important task of all trade union committees. They must help those competing for the title of teams and shock-workers of Communist Labor to develop socialist responsibilities directed toward the furthest possible growth of productivity of labor, raising the quality of production and reducing its cost, mastering new technique, improvement of the technology of production, automation and mechanization of productive processes, raising the technical and general educational level.

The trade union committees should obtain from the factory executives the creation for the competitors of all the necessary productive conditions for fulfillment of the obligations they have undertaken, and also should render them help in increasing general educational and technical knowledge.

It is necessary to organize regular study and extensive distribution of the work experience of teams and shock-workers of Communist Labor, to issue brochures and placards systematically, to organize speeches of partici-

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* Coefficient of utilization of the useful volume of the furnace.

pants in the competition in the press, on the radio, and on television.

The creative cooperation of scientific workers with competitors for the title of teams and shock-workers of Communist Labor, which originated in enterprises of the ferrous metal industry in the Sverdlovsk, Stalinsk, and Dnepropetrovsk districts, deserves imitation. It is necessary to develop this cooperation even further, to attract the scholarly and scientific workers of the institutes, and engineers and technicians, more extensively to leadership of those competing for the title of teams and shock-workers of Communist Labor.

Our country has entered a period of widespread building of a communist society. The start of this new period can be seen everywhere, but is is especially manifest in the work of the collectives which have already decided to live and work in the communist way. These private soldiers in the great army of labor have decided to be the first to seek out and struggle along the road to communism.

To render every sort of help to these collectives in their noble endeavor is the sacred duty of every trade union committee in the enterprises of the ferrous metals industry.

* * *

SMELTING BLAST FURNACE FERROSILICON WITH OXYGEN-ENRICHED BLAST

Yu. A. Popov, E. A. Gamayunov, and A. T. Filippov

Chelyabinsk Steel Plant

Since February 1958 at one of the Chelyabinsk Blast Furnaces, ferrosilicon has been smelted with an oxygen-enriched blast. Experience has shown that each percent increase in the oxygen content in the blast, with a total increase in the oxygen content to 25%, increases the furnace output by 4.0-4.5% and reduces the consumption of coke by 0.9-1.5%. The technical and economic data for blast-furnace smelting during 1958 are given in Table 1. With an average content of oxygen in the blast of 23.5%, the productivity of the furnace for the smelting of ferrosilicon in 1958 increased by 17% compared with 1957, and by 14.3% in comparison with 1956; the coke consumption was reduced correspondingly by 6.0 and 1.8%. The deterioration in furnace operation in 1957 was due to uneven working as the result of strong growth of incrustation in the shaft.

During the introduction of the oxygen-enriched blast into the blast furnace, a brief experimental smelting of ferrosilicon was carried out with reduced amount of metal additions in the charge (120 kg/ton iron instead of the usual 400 kg/ton). The smelting gave relatively good economic and technical results.

One of the drawbacks in smelting ferrosilicon in the Chelyabinsk blast furnaces is the systematic formation of incrustations in the upper part of the shaft of the furnace (Fig. 1). The annular incrustations spoil the operation of the furnace, increase the entrainment of dust and reduce the output.

Before the blast furnace was changed over to oxygen blast, the incrustation cracked together with the lining. However, since the shell of the shaft was soaked with water, the incrustation again stuck to it, forming a kind of self-generating lining. With this lining the furnace operated for 5 months, during 3 of which ferrosilicon was smelted.

During the introduction of ferrosilicon smelting with enriched blast, samples were taken of the smelting materials from the hearth through a tuyere and the temperature and the composition of the gas were measured along the radii of the hearth and the throat of the blast furnace.

The changes in temperature and composition of the gas at the level of the tuyeres during the smelting of ferrosilicon are shown in Fig. 2. The maximum temperature was at a distance of 750 mm from the eye of the tuyere and was 1980-2080°; closer to the axis of the

TABLE 1. Technical and Economic Data for the Smelting

Indices	Years			Months 1958							
	1956	1957	1958	January	February	March	April 1–29	April 11–19	April 20–29	June	July
Coefficient of utlilization of useful vol	0.562	0.576	0.495	0.559	0.521	0.503	0.471	0.480	0.460	0.463	0.484
Coke consumption, ton/ton iron	1.159	1.208	1.129	1.158	1.156	1.115	1.131	1.146	1.100	1.130	1.134
Yield of slag, ton/ton iron	0.590	0.534	0.514	0.608	0.631	0.573	3.532	0.530	0.470	0.476	0.435
Dust entrainment, ton/ton iron	0.371	0.552	0.279	0.160	0.170	0.220	0.228	0.236	0.215	0.207	0.336
Intensity of combustion of coke, kg/m³. day	821	850	955	860	909	926	969	950	957	955	921
Consumption of metal additions, ton/ton iron	0.454	0.464	0.446	0.430	0.390	0.430	0.363	0.120	0.442	0.480	0.540
Content of oxygen in blast, %	21.0	21.0	23.5	21.0	23.2	24.5	25.1	25.0	25.2	25.9	25.5
Si content in iron, %	10.44	10.12	10.23	10.19	10.39	10.31	11.03	10.75	10.70	10.17	9.94
Composition of slag, %											
CaO	30.66	32.12	29.03	31.89	32.79	32.28	31.85	30.97	30.49	30.14	29.72
SiO_2	45.53	44.73	44.81	46.00	43.68	43.61	43.40	45.00	40.83	46.28	46.10
Increase in output compared with January, 1958, %	—	—	—	—	7.1	11.0	16.7	17.2	22.00	19.3	15,2
Reduction in consumption of coke compared with January, 1958, %	—	—	—	—	—	3.7	2.7	1.1	4.2	2.4	2.07

TABLE 2. Chemical Composition of Iron and Slag Taken Along the Radius of the Hearth Through the Tuyere and at the Tapping Holes

Place sample taken	Composition of iron %					Composition of slag %						
	Si	Mn	S	P	C	SiO_2	Al_2O_3	CaO	MgO	MnO	FeO	S
At a distance of 2 500 mm from the eye of the tuyere	12.9	1.04	0.013	0.14	1.05	40.62	18.22	32.65	6.23	0.44	0.62	0.86
From the trough	10.8	1.9	0.040	0.16	2.05	42.98	16.92	30.82	6.78	0.65	0.10	1.14
At distance of 1100 mm from the eye of the tuyere	6.42	2.30	0.049	—	—	35.45	18.82	32.23	5.29	0.35	2.0	—
From the trough	10.52	1.8	0.022	—	—	41.80	17.50	33.57	6.30	0.66	0.12	0.76
At a distance of 100 mm from the eye of the tuyere	9.87	1.25	0.068	0.14	1.74	—	—	—	—	—	—	—
From the trough	9.8	1.65	0.034	0.15	1.98	—	—	—	—	—	—	—

furnace it gradually reduced and in the center was 1300-1500°. The temperatures were measured at 1-3 points using a tungsten-molybdenum thermocouple with a molybdenum tip, which was introduced into the furnace through a cooled pipe of 56 mm diameter.

The comparatively small content of carbon monoxide in the furnace gas in the central part of the furnace (38% on ordinary blast and 48% with enrichment up to 25%) indicates that the reduction of primary materials below the level of the tuyeres is very weakly developed; this is also confirmed by the composition of the iron and slag taken at this level (Table 2).

On the basis of the material and thermal balances of the smelting it was found that with increase in oxygen in the blast to 25%, the degree of direct reduction of iron is reduced from 50-55 to 37-48%.

From the data of Table 2, taking into account the temperature and composition of the gases, it can be concluded that the silicon in the blast furnace is mainly

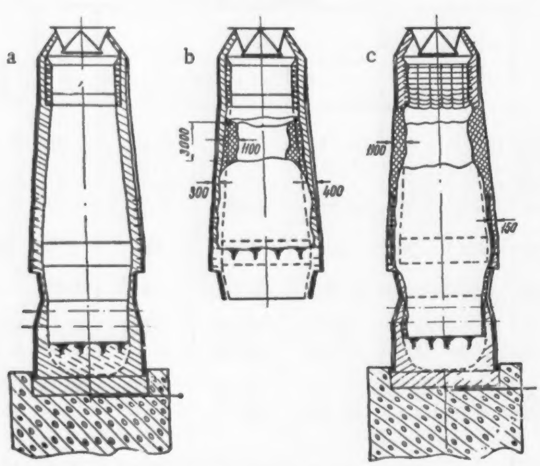


Fig. 1. Incrustations in blast furnace: a) profile after middle overhaul (August 1956); b) creeping of incrustation; c) profile before middle overhaul (May 1958).

reduced above the level of the tuyeres. In a reducing atmosphere at a distance of 2500 mm from the eye of the tuyere and at a temperature of 1400°, the silicon content was 12.9%—somewhat higher than in the final metal.

In the oxygen region of the tuyere zone at a distance of 1000-1200 mm from the eye of the tuyere, the silicon, iron and other impurities are partially oxidized. The content of silicon in the oxygen region was 6.7-9.8%, and in the center of the furnace, as already mentioned—12.9% (although at temperatures of 1800-2080° in the tuyere zone the content of silicon should be much higher).

The results of the investigation of the gas composition along the cross section of the throat of the furnace during the smelting of ferrosilicon with ordinary and with oxygen blast are given in Fig. 3.

Due to the systematic growth of incrustations above the shaft and the uneven operation of the furnace, it was not possible to determine the accurate dependence of the temperature and character of the gas stream along the height of the furnace on the degree of enrichment of the blast with oxygen.

It was found, however, that when the blast was enriched with up to 25% oxygen, the content of CO_2 in the throat gas increased from 5-7 to 7-10% and the temperature of the throat gases was reduced. To keep the temperature of the blast-furnace gases within the limits of 200-300°C, the flow of water to cool the throat (with an oxygen content in the blast of 24-25%) was halved; in some periods the supply of water at the throat was completely stopped.

Zonal thermal balances were compiled showing the change in the thermal state at various levels for different degrees of oxygen enrichment of the blast. In January, 1958, when the furnace worked on atmospheric blast, 955 kg of coke carbon were used per 1 ton of iron, 747 kg being burnt at the tuyeres. The yeild of gas was 4480 m^3/ton ferrosilicon. In April, 1958, the blast contained 25.1% O_2. The consumption of coke carbon was reduced to 865 kg/ton, of which 682 kg/ton was burnt at the

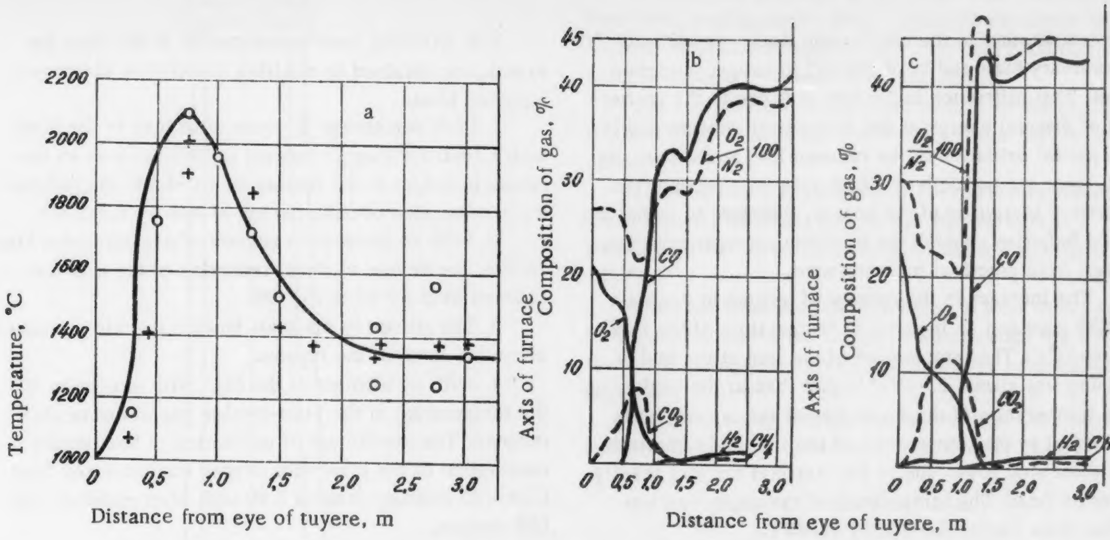


Fig. 2. Change in temperature and composition of the gas along the radius of the hearth at the level of the tuyeres: a) temperature of gas; b) composition of gas with atmospheric blast; c) composition of gas with an oxygen content in the blast of 23.2%; +) at atmospheric blast; o) with an oxygen content in the blast of 24-25%.

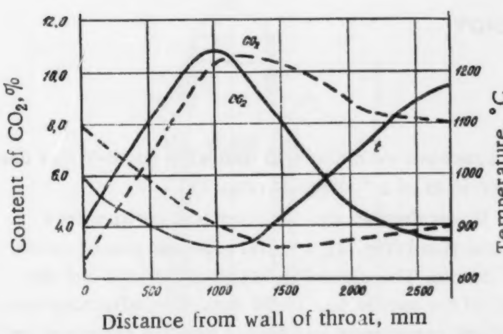


Fig. 3. Change in the content of CO_2 and the temperatures of the blast-furnace gas across the radius of the throat; -----in ordinary blast; ——with enriched blast.

tuyeres; the yield of gas was 3970 m^3/ton. This fall in the consumption of coke was due to the decrease in the degree of direct reduction and improvement in the use of the heat of the gas.

From the 20th to 29th of April, the number of metal additions to the furnace was increased, leading to reductions in the amount of slag and a still further reduction in the consumption of coke carbon (to 835 kg/ton, 655 kg/ton being burnt at the tuyeres). The yield of gas was then 3045 m^3/ton.

Figure 4 shows the change in the coefficient of utilization of the heat of the gas at various levels of the

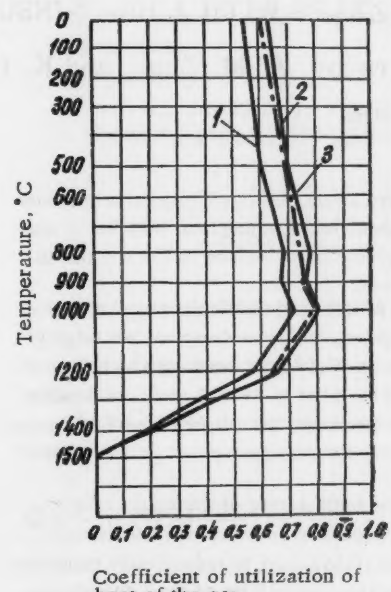


Fig. 4. Change in the coefficient of utilization of heat along the height of the furnace, in relation to the temperature: 1) January, 1958, atmospheric blast; 2) April, 1958; 3) April 19-28, 1958.

furnace. This coefficient is equal to the ratio of the amount of heat necessary for all essential processes at a given temperature, to the heat content of the gas. The greatest evolution of heat occurs at 800-1000°C. It can be seen from Fig. 4 that with reduction in the yield of gas its heat evolution increases.

The theoretical temperature of combustion in ordinary blast, according to calculation, should be 2200°C, in oxygen-enriched blast it should be 2400°C. In actual fact,

the temperature in the combustion flare was 1980-2000°C in ordinary blast and 2020-2080°C in oxygen-enriched blast. This difference in the results is due to the scattering of thermal energy at the moment of ignition and to the partial oxidation of the reduced iron and silicon. As yet, there are no methods which take into account the degree of scattering of the energy, therefore in compiling zonal balances we took the maximum temperature determined from practical measurements.

The increase in the content of oxygen in the blast to 25% provided an increase in temperature of the iron of 30-35°C. The temperature of the iron at the end of tapping was always 20-30°C higher than at the beginning. The temperature of the lower slag at the tapping holes was equal to the temperature of the iron and sometimes was somewhat lower due to the fact that the slag readily gives off heat. The temperature of the upper slag was higher than that of the iron by 40-80°C.

The following conclusions can be drawn from the experience obtained in smelting ferrosilicon in oxygen-enriched blast.

1. Each percentage increase of oxygen in the blast with a total increase in content to 25% leads to an increase in output of the furnace to 4.0-4.5% and reduces the consumption of coke per ton of iron by 0.9-1.5%.

2. With an increase in content of oxygen in the blast to 25%, the degree of direct reduction of the iron was reduced from 50-55 to 37-48%.

3. The silicon in the blast furnace is mainly reduced above the level of the tuyeres.

4. With enrichment of the blast with oxygen to 25%, the temperature of the blast-furnace gas was somewhat reduced. The coefficient of utilization of heat during enrichment of the blast with oxygen was increased from 0.54 with ordinary blast to 0.62 with blast enriched with 25% oxygen.

* * *

TUYERE NOZZLES WITH A HEAT-INSULATING COATING

V. I. Surovov, A. M. Zhak, and K. I. Kotov

Petrovskii Plant

At temperatures above 800°C, ordinary steel nozzles warp and rapidly break down, much heat then being lost by radiation through the outer surface of the nozzles to the surrounding space.

The screened double-walled nozzles suggested by workers at the Magnitogorsk Steel Combine are largely free from these faults. They have been used to increase the temperature of the blast to 1000°C and considerably reduce heat losses. However, the durability of the screened nozzles under the conditions of our plant was very low (1-3 months).

To increase the temperature of the blast furnaces above 900°C, to reduce heat losses, to save the scarce heat-resistant steels (ÉI652, etc), to reduce coke consumption and to increase the productivity of the furnaces, Korobov and Surovov manufactured and introduced nozzles with a heat-insulating coating.

The coating was prepared in the following way.

Foamed chamotte and periclase-spinel were ground to a coarseness of 2 mm and mixed in the proportions 85:15. The mixture was carefully stirred and about 15-20% by weight of molten glass was added to the powder. The excess of molten glass should not separate from the mixture under the action of the rammer.

An ordinary steel-cast nozzle was bored from the inside along the edges; at a distance of 40 mm from its edge, recesses were made 25 mm wide and 5-7 mm deep in the form of a "dovetail" (Fig. 1).

It was then set up in the vertical position on a special stand (Fig. 2); a metal pipe was placed inside the nozzle. The clearance between the pipe and the wall of the nozzle was 18-20 mm. The refractory mass of thick consistency was poured into the clearance and firmly rammed by special narrow rammers. The pipe was then removed and the nozzle dried in air for a day. After this it was placed in a furnace and the temperature raised at the rate of 100°C/hr to 1050°C. The nozzle was kept at this temperature for 1 hour and then cooled together with the furnace for 8 hours. The grooves in the form of a "dovetail" were filled when the refractory mass was rammed and prevented the lining of the nozzle from moving along the axis. A better quality ramming was obtained if the nozzle was heated together with the inserted tube. The pipe should be carefully wrapped with sheet iron, otherwise the refractory mass can stick so firmly to the metal of the pipe that it cannot be removed; the pipe can readily be removed from the iron, the inner surface of the nozzle then remaining smooth and even.

After annealing, the nozzles were taken to the machine shop where warping and other defects were removed, after which the nozzle was placed in the furnace.

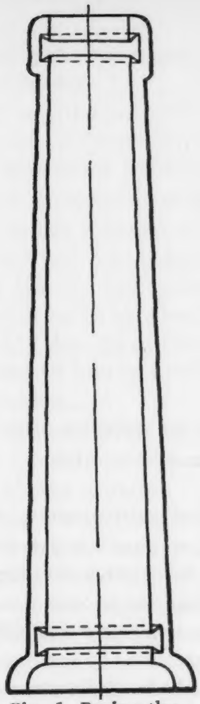


Fig. 1. Boring the nozzle before ramming.

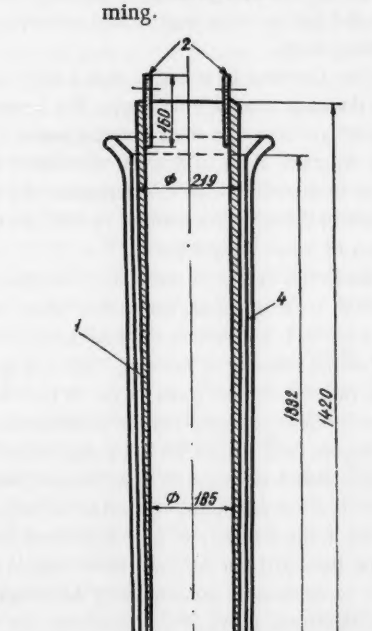


Fig. 2. Setting the nozzle on the stand: 1) pipe; 2) "lugs" for drawing out pipe; 3) stand; 4) nozzle.

The nozzles installed at the three blast furnaces have been working since May, 1959 and have shown no defects. With blast temperatures of 900 and 1000°C, the outer surfaces of the nozzles do not glow.

The temperature of the outer surface of the nozzles is 346-470°C; in all cases the surface of the nozzle is dark.

The resistance to crushing of the refractory samples taken directly from the nozzle was 90-100 kg/cm^2, porosity 38.3%, volumetric weight 1.58, refractoriness 1650°C.

The heat-insulating mass adheres so well to the walls of the nozzle that when slag is poured through the nozzles they can be cleaned without damaging the heat-insulating layer.

The heat losses through the hot walls of the nozzle (without lining) are determined from the formula:

$$Q=4Fk\left[\left(\frac{T_1}{100}\right)^4-\left(\frac{T_2}{100}\right)^4\right],$$

where Q is the quantity of heat, kcal/hr;

4 is the coefficient of radiation, kcal/m^2 × deg^4 × hr;

F is the surface of the nozzle, m^2;

T_1 is the absolute mean temperature of the nozzle surface;

T_2 is the absolute mean temperature of the surrounding medium;

k is the number of nozzles.

For k=12, F=1.02m^2, T_1=800+273=1073°, T_2=27+273=300°; the amount of radiated heat is:

$$Q=4\cdot 1.02\cdot 12\left[\left(\frac{1073}{100}\right)^4-\left(\frac{300}{100}\right)^4\right]= = 645000 \text{ kcal/hr},$$

which corresponds to 645000/3700=174 kg coke per hour (3700 kcal is the amount of heat acquired by the blast furnace on the combustion of 1 kg of coke), or about 4 kg/ton iron.

The heat losses through the heated nozzles with a heat-insulating layer are:

$$Q=4\cdot 1{,}02\cdot 12\left[\left(\frac{743}{100}\right)^4-\left(\frac{300}{100}\right)^4\right]=142300 \text{ kcal},$$

which corresponds to 38 kg coke per hour, or 0.9 kg/ton iron.

The saving in coke by using nozzles with a heat-insulating lining, due to reduction in the heat losses, is therefore (4-0.9)=3.1 kg/ton iron. Due to this reduction alone, the annual saving at the Petrovskii Plant was about 800,000 rubles. Furthermore, nozzles with heat-insulating layers are cheaper than nozzles of heat-resistant steel. The saving resulting from the replacement of steel nozzles by lined nozzles is greater than 300,000 rubles.

The total saving is therefore 1,100,000 rubles.

* * *

ON THE REDUCTION OF THE SULFUR CONTENT IN PIG IRON

M. A. Shapovalov

Central Scientific Research Institute for Ferrous Metallurgy

The journal Metallurgist, No. 1 (1959), contained an article by I. I. Korobov, "On the reduction of the sulfur content in pig iron during blast-furnace smelting." * Below are published the responses to this article.

The presence of sulfur in steel has an extreme negative effect on the properties of metal. Combating this problem is practically impossible in open-hearth furnaces and converters due to the high content of ferrous oxide in the slags. The primary removal of sulfur from the metal therefore occurs during blast-furnace production. Here, desulfurization occurs successfully, although it very greatly hinders operation of blast furnaces and considerably reduces their indices.

For a greater desulfurization of pig iron (the necessity for this steadily increases) it is necessary that basic slag be heated to such a degree that only traces of ferrous oxide remain in it. Even an insignificant content—tenths of a percent—severely reduces the desulfurizing power of slag. In the tuyere zone where the temperature is maximum the ferrous oxide content in slag is 20-30%, which has a negative effect on desulfurization in this zone of the furnace. The most active desulfurization occurs below the tuyere where the content of ferrous oxide in the slag is minimum. However, the height of this zone is very limited, and the short duration of contact between the pig iron and the slag must be compensated for by their elevated heating.

If desulfurization is conducted outside the blast furnace, then it is possible to smelt with acid slags, which makes it possible to solve a number great problems of blast-furnace production and to improve substantially the technical-economic indices of blast-furnace smelting.

Therefore, I. I. Korobov's remarks on the pages of the journal Metallurgist where the author denies the positive value and rejects even the very possibility of conducting blast-furnace smelting in acid slags seems somewhat strange. It is impossible to agree with his assertion that cooling of the furnace hearth with all resultant consequences is inevitable when smelting with acid slags.

I. I. Korobov's arguments can be acknowledged as true only as applied to smelting an unprepared charge with a weakly developed indirect reduction of iron, with a high development of direct iron reduction, and with great heat consumption in the furnace hearth. Actually, under these conditions the conversion of a furnace to acid slags will not yield positive results. Presently many blast furnaces operate on a well-prepared charge and with natural gas, which in conjunction with correct management of smelting assures a high development of the indirect reduction of the iron (more than 65%): Direct reduction is almost completed on the boundary of the boshes and the furnace body. Under these conditions the requirement for heat in the hearth will be minimum, and this makes it possible to carry out blast-furnace smelting at a low heat and temperature regime and consequently with low-melting slags.

I. I. Korobov is wrong in asserting that a low content of alumina in the slags (below 6 and even 5%) causes an approximate 100° reduction in their melting point. On the melting-point diagram, slags with a low content of alumina are arranged on isotherms of high temperatures and those with an increase in the alumina content to 20% are shifted to the region of lower temperatures.

An increase in the height of the zone of temperatures above 950°, which is, so to speak, inevitable when smelting with acid slags as I. I. Korobov indicated, can occur independently of the basicity of the slag, but only in the case where the ratio of the hot flows of gas in the charge with respect to the heat consumption for endothermic reactions is increased. Such a case for acid-slag smelting is excluded since with a decrease in heat consumption in the hearth there is a corresponding reduction in both coke consumption and in the quantity of gas. Therefore the ratio of the heat flow will not only not be increased but conversely will be decreased; consequently the height of the zone of temperatures above 950° is reduced, the admission of heat into the shaft is decreased, and the temperature of the blast-furnace gas is reduced, which is attested to from the operational experience of blast furnaces of, for example, the Cherepovets Plant.

A decrease in the amount of sulfur in the charge by lower coke consumption and a lower sulfur content in it nevertheless requires maintaining in the blast furnace a high basicity of the slag and its considerable superheating,

* See English translation.

which is confirmed by the operational practice of the Magnitogorsk blast furnaces.

I. I. Korobov's assertion was also erroneous that supplying oxygen to the furnace hearth will make it possible to increase the slag temperature and thus to improve pig iron desulfurization. A supply of oxygen to the hearth will not increase the slag temperature without the additional consumption of fuel. When oxygen is supplied to liquid pig iron the iron will burn down with the liberation of heat, which will cause an increase in the slag temperature. Simultaneously, the slag will be saturated with ferrous oxide which will lead to a worsening of the desulfurization conditions.

I. I. Korobov's assertion that cooling of the hearth is inevitable with acid slags is disproved by the operational experience of blast furnaces.

It is known that at blast furnaces at the Kuznets and Nizhne-Tagil' Combines the basicity of the slag is 1.0, and heating of the pig iron is sufficiently great.

At the furnaces of the Magnitogorsk Combine when operating with relatively acid slags with the ratio of $CaO: SiO_2=1.1$, the temperature of the pig iron reaches 1500°, and the slag 1600°. Such a high temperature is maintained only for increasing the degree of desulfurization of the pig iron, although it has a negative effect on the service of the refractory work as well and causes excessive reduction of the silicon.

The Cherepovets Plant smelts with a slag ratio of $CaO: SiO_2=1.10$. A further reduction of basicity is prevented not by the low temperature of slag smelting, but by the increase in the viscosity of the slag due to the low content of Al_2O_3 and MgO in it. With an increase in the content of these components there would unconditionally be a decrease in the basicity to 1.

In the overwhelming majority of cases the basicity of the slag was considerably below unity in charcoal blast-furnace smeltings. Their heat content is about 380 kcal/kg and melting point below 1400°. Heating of the blast was low, the volume of the hearth was small. All blast furnaces operated normally. Consequently, the presence in the slag of 6-7% Al_2O_3 (which is found everywhere in the southern plants) and 4-5% MgO, with large size furnaces and high heating of the blast, makes it possible to conduct blast-furnace smelting with more acidic slags having a ratio $(CaO+MgO): SiO_2$ approximately equal to unity.

What can be expected from the transfer to more acidic slags?

In modern steel melting production, especially with the use of an oxygen blast, a tendency toward a reduction in the silicon content to the minimum possible magnitude has definitely been noticed in conversion pig iron. The content of silicon in pig iron of 0.6-0.9% is considered as too great, since during conversion of such pig iron the amount of slag increases and the duration of open-hearth smelting is prolonged. Due to this, ways are being sought to desulfurize pig iron outside the blast furnace prior to teeming in steelmaking units. Thus an apparent contradiction is obtained. In a blast furnace a great quantity of heat is consumed on the reduction of silicon, but in steelmaking production silicon is not only not necessary but is harmful. Consequently, it is necessary to manage blast-furnace smelting so that pig iron is produced with a minimum content of silicon; this will be advantageous for blast-furnace and steelmaking production. Such smelting is possible, which is confirmed by the experience of smelting Thomas pig iron with a content of silicon of about 0.3%. Low-melting slags, especially in conjunction with blasting natural gas to the furnace hearth, is one of the main conditions for producing low-silicon pig iron.

Blasting of a reducing gas decreases the consumption of coke, but as a result the gas-dynamic conditions of blast-furnace smelting are worsened since in this case the quantity of gas per unit of coke is increased. In order not to allow a worsening of the gas-dynamic conditions it is necessary to improve the gas permeability of the charge materials both by reducing the content of minor fractions in the skip coke and by increasing the strength of the fluxing agglomerate. Moreover it is known that an increase in the lime content in the agglomerate above a certain magnitude somewhat decreases its strength. The use of more acidic slags makes it possible to produce a complete fluxing agglomerate with a lower content of lime, which excludes the need for delivering raw limestone to the charge. Thus, with a basicity of 1.25 and a content of 15% SiO_2 in the ore concentrate there will be 18.75% CaO in the agglomerate, while with a basicity of 1 there is only 15%, i.e., 1.25 times less. Here the quantity of CaO is decreased by 37.5 kg/ton of the agglomerate, the yield of slag is reduced by 75 kg/ton of pig iron, and the content of Al_2O_3 in the slag is increased. All these factors improve the gas-dynamic conditions of blast-furnace smelting. Thus, the transfer to more acidic slags considerably facilitates the production of low-silicon pig iron and of a strong, completely fluxing agglomerate even in the case where this problem is now difficult to solve.

Up to the present time much attention has been devoted to the chemical side of desulfurization. Unfortunately a reliable method and a satisfactory design of units for desulfurization outside the blast furnace have not been developed, while it is just these that are the main factors in the problem being examined. It is necessary to hope that in the future such a unit will be developed. Removing the desulfurization process from the blast furnace will make it possible to transfer to more acidic slags with a ratio of $(CaO+MgO): SiO_2=1$, which will make it possible:

1) to conduct smelting at a maximum low heat and temperature regime, facilitating the assembly of the charges and the service conditions of the refractory work of the hearth;

2) to smelt low-silicon pig iron;

3) to use a completely fluxing agglomerate with a low content of lime and consequently a stronger agglomerate;

4) to reduce the consumption of lime and yield of slag, and also to increase the content of alumina in the blast furnace slag.

* * *

I. S. Kulikov

Institute of Metallurgy, Academy of Sciences of the USSR

The problem of optimum slags for blast-furnace smelting is one of the least clear problems in the theory of the blast-furnace process. The journal Metallurgist has published several articles devoted to the selection of the composition of blast-furnace slags necessary for producing low-sulfur pig iron.

Two opinions were expressed on the question of conducting the blast-furnace process. I. I. Korobov considers that desulfurization of the pig iron in the blast furnaces can be attained by means of a good preparation of the materials and the development of slags of low basicity under conditions of an increase in the hearth temperature.

G. A. Volovik and M. A. Shapovalov think that it is necessary to operate on acid slags with subsequent desulfurization of the pig iron outside the blast furnace. They assert that operation with acid slags due to a decrease in the amount of slag should be accompanied by an increase in the productivity of the blast furnaces and a decrease in the specific consumption of coke. This opinion does not agree with the experience of blast furnace operation, which I. I. Korobov indicates, nor with the smelting experience at the Magnitogorsk Metallurgical Combine, on which data, unfortunately has not been published.

It is difficult not to agree with the considerations of I. I. Korobov on the inexpediency of operating blast furnaces on acid slags. Actually, in order to ensure an even and forced rate of the blast furnaces it is necessary to have non-viscous slags. Acid slags, even severely superheated, have high viscosity, yield a zone of the solid-liquid state higher in the furnace filling the cavities between the coke pieces, and offer great resistance to the passage of gases. Therefore operation with acid slags cannot be as forced as with neutral or slightly basic slags.

The citations of G. A. Volovik and M. A. Shapovalov on the successful operation of charcoal furnaces on acid slags cannot be acknowledged as well grounded. The aerodynamic conditions of operating modern blast furnace and charcoal furnaces are completely different. Charcoal is more porous than coke; it occupies more than 90% of the space filled by the charge and therefore the physical properties of the slags (particularly the viscosity) do not have a substantial effect on the gas permeability of the column of charge materials. In addition to this, modern blast furnaces operate on a more forced regime, i.e., with greater gas velocities than charcoal furnaces, and the resistance of the slag-formation zones in them, depending on the physical properties of the slags, plays an undoubtedly greater role.

The specific consumption of coke with a given composition of slag increases with an increase in the yield of slag. However, acid slags as a consequence of earlier slag formation slow down the reduction processes and cool the furnace hearth, since they enter into it more heated because the degree of direct reduction and consequently the expenditure of heat in the hearth is greater than when operating on basic slags. The decrease in the slag yield therefore cannot compensate for the loss of heat or for the increase of direct reduction and cooling of the hearth.

An analysis of the behavior of sulfur in the hearth of a blast furnace shows that in order to reduce the content of sulfur in pig iron it is necessary:

1. to operate on a slag with a sufficiently high desulfurizing power;

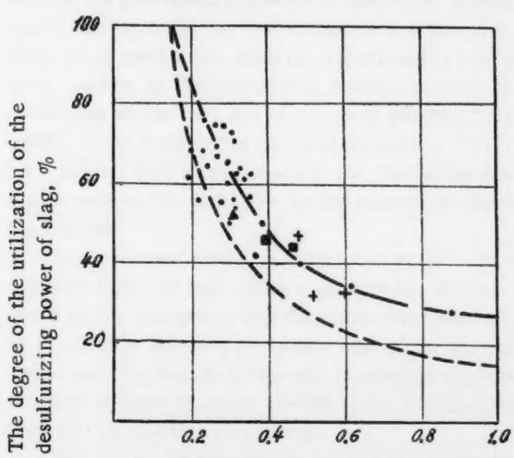
2. to create in the blast-furnace hearth conditions which ensure the maximum use of the desulfurizing power of slag.

The second condition is no less important than the first. An investigation of the equilibrium of desulfurization in the system pig iron-slag-carbon-gas phase and an analysis of the operation of blast furnaces showed that equilibrium of desulfurization in blast furnaces is not attained and that the slowest stage of the process is the stage of slag deoxidation. The content of ferrous oxide in the slag (figure) can therefore be taken as the index of the degree of utilization of the slag desulfurizing power. It is seen from the figure that the lower the final content of ferrous oxide in the slag the better the desulfurizing power is used. Thus in 1951 at the Magnitogorsk Metal-

lurgical Combine the content of ferrous oxide was 0.8% and only 25-30% of the slag desulfurizing power was used. In 1957-1958 the ferrous oxide content dropped to 0.28% and the use of the desulfurizing power of the slags increased to 60-70% (the bottom slag). This was attained as a result of taking a number of measures: conversion to fluxing agglomerates, removal of manganese ores from the charge with a certain increase in magnesium oxide in the slags, and an increase in the pressure and heating of the blast. All this made it possible to increase the degree of reducing iron oxides to the moment of the start of slag formation, and consequently to reduce the content of iron oxides in the upper slag. The smelting experience at the Magnitogorsk Metallurgical Combine showed the possiblity of a further increase in the use of the desulfurizing power of slag. The average monthly content of ferrous oxide in the lower slag was 0.17%; the use of the desulfurizing power of the lower slag exceeded 90%, and for all slag was about 70%.

This was produced mainly by a decrease in the viscosity of the slags with an increase in magnesium oxide from 8.5 to 10-12%.

The example cited shows that as a result of improving the technology of blast-furnace smelting it is possible to increase not only its technical-economical indices for productivity and coke consumption, but also by a better use of the slag, to solve the problem of producing low-sulfur pig iron. In order to force the rate of the blast furnaces and to maintain in the furnace hearth a temperature necessary for the successful completion of the physical-chemical processes it is necessary to have slags with a high fluidity and a temperature of crystallization of 1300-1350°; that is, slags with a ratio of CaO: SiO_2= 1.05-1.15. The viscosity of such slags in the absence of manganese oxide and iron oxide at a temperature of 1500 -1550° does not exceed 5 poise. With an increase in the content of alumina and magnesium oxide to 10-15% at the same temperatures it is reduced to 2-3 poise. These slags permit a more forced operation of the furnaces than the acid or too-basic slags. In addition to this, with a content of 8-12% Al_2O_3 and about 10% MgO, it is possible to refrain from additions of manganese ores, to raise the ratio of CaO: SiO_2 to 1.2, and even under conditions in the South, to smelt low-sulfur pig iron.



The degree of utilization of the slag desulfurizing power as a function of the content of ferrous oxide in it: ● Magnitogorsk Metallurgical Combine, 1951-1957; ▲ Chelyabinsk Metallurgical Plant, 1957; ■ Dzenzhinskii Plant; + after the data of V. E. Vasil'ev, 1932.

It is necessary to note that the desulfurizing power of slags is virtually independent of the temperature and therefore an increase in the degree of its utilization is advantageously attained without an increase in the hearth temperature.

It is impossible to smelt out low-sulfur conversion pig iron from a sulfurous charge with acid slags because an increase in the temperature inevitably entails a sharp increase in the silicon content and naturally will not give coke economy. Therefore the blowing of oxygen to the center of the furnace hearth apparently is not advantageous. Likewise the blowing of lime through the slag tapping hole is not advantageous since this leads only to a local increase in the basicity of the slag and disrupts the uniformity of the furnace operation.

Finally it is necessary to note the positive role of the silicon of pig iron as a deoxidizer of slag, accelerating the desulfurizing process. Silicon is reduced mainly above the level of the tuyeres. In the zone of the tuyeres, in the non-oxidizing zone where the temperature reaches the maximum, the degree of silicon reduction and the silicon content in the pig iron are also the greatest. When the drops of pig iron pass through the layer of slag with a lower temperature than at the tuyeres the silicon begins to be oxidized, reducing the iron and manganese from the slag. This favors a more complete reduction of the slag and a further desulfurization of the pig iron.

In our opinion desulfurization outside the blast furnace should not be recommended for ordinary conversion pig iron because even under conditions in the South, with slags ensuring the best technical-economic indices for production and coke consumption, the pig iron will be conditional for sulfur. Moreover, not only in the Urals and Siberia, but also in the South USSR it is possible to organize the blast-furnace process in such a way that it is possible to smelt low-sulfur, low-manganese conversion pig iron with a sulfur content to 0.03% without a substantial increase in the production cost of the pig iron and without a decrease in productivity. For this purpose it is necessary to improve the technology of blast-furnace smelting and to ensure the plants of the South a quality dolomitized limestone or dolomite.

Desulfurization of pig iron outside the blast furnace can be economically justified only when operating on a rich high-sulfur, low-phosphorus charge providing that

sulfurous pig iron is smelted with slags having optimum properties and it is desulfurized by lime, for example, in rotary furnaces whose capacities assure reception of the entire output (here overflows of pig iron are excluded).

Such a technological arrangement at the present time still has not been developed; construction of a test installation and testing is needed.

It is necessary to emphasize here the urgent need for an over-all decrease in sulfur in the charge materials of blast-furnace smelting at all stages of their preparation.

In concluding the article it is necessary to note that the reserves for improving desulfurization in blast furnaces are still very great. The conditions of their utilization are the preparation of the materials for blast-furnace smelting, the creation of such conditions in the furnace so that reduction is completed prior to the start of slag formation, and the production of slags with optimum physical and chemical properties. When observing these conditions together with a significant decrease in the sulfur content in pig iron the furnace productivity will undoubtedly increase and the coke consumption will decrease. The economy for all USSR plants, which can be obtained in this case, consists of many millions of rubles.

* * *

NEW DEVELOPMENTS FOR THE COMPLEX AUTOMATION OF ARC FURNACES

Yu. E. Efroimovich, V. M. Vinogradov, A. F. Kablukovskii, N. I. Shutkin, N. I. Gorbatov, and T. I. Zuev

Central Laboratory for Automatics, "Élektrostal'" Plant

In line with the problem of complex mechanization and automation of the process for melting steel in arc furnaces at the "Elektrostal'" Plant, work has been carried out on the improvement of the design elements of the furnace. New devices have been developed for checking and controlling the most important features of the melting process, and investigations have been carried out on the electrical and thermal operation of the furnaces and their connection with the metallurgical process of melting*.

Improving the design elements of the furnace. Both in the USSR and abroad, a serious fault in the designs of current carriers from the furnace transformer to the electrodes (short connections) is the inequality and high values of the phase resistances. The result of this is that in furnaces of higher capacity than 10-15 tons, all the operational features of the furnaces are reduced. Usually in the designs of the short connections from the 6 outlets of the transformer (the beginnings and ends of its windings) to the electrodes there are 3 assemblies of busbars, cables and pipes. In the new design used at the plant, there are 6 assemblies, each of approximately half the cross section. By this means it is possible to connect assemblies at the start and end of each winding from the outlets of the transformer to the electrodes in series, i.e., bifilar. This arrangement of the assemblies considerably reduces the resistance to the passage of alternating current.

The new short circuit is introduced at furnaces of capacity 5 and 20 tons. Their reactance is 1.6-2.8 times less than the old system and the electrical losses are less by about 20%. Whereas previously with equal currents in the phases, the powers of the arcs in average capacity furnaces differed by about 15-20%, with the new system they will be practially identical.

The use of the new system has led to a reduction in melting time in both furnaces and to a reduction in the specific consumption of electrical power, and at the 20-ton furnace the power factor ($\cos\varphi$) has been increased by more than 0.05. At the present time, these results are being checked for a large number of melts.

The new system should be used on new and on existing furnaces. Its use at the large furnaces of the "Dneprospetsstal'" Plant with existing transformers of about 15000 kv-a has reduced the melting time by 30 minutes and increased $\cos\varphi$ by 0.05.

It is a well-known fact that in the electrode holder, at the point of contact between the jaw and the electrode, there is a considerable transitory resistance, which changes during the melting and causes considerable losses in electrical power (up to 1.0%). In order to reduce this resistance and to eliminate the causes for changes in it, a new design was developed for a water-cooled electrode holder, made of a non-magnetic material. In the new electrode holder the current is fed from two opposite sides by two contact surfaces. At the surface of the jaws touching the electrodes, a layer of chrome copper is welded. This more than halves the contact resistance.

At a number of furnaces the movement mechanisms have been redesigned and at one of them the balancing of the electrode has been changed. At most of the existing furnaces the electrodes are moved by a motor through a reduction gear consisting of two worm transmissions and a drum. By means of cables, the drum moves the telescopic stand of the electrode holder. The system must have an "idle" stroke so that for any change in the direction of rotation of the motor shaft, the electrode begins to move, not immediately but only after a certain time during which the shaft of the motor completes up to 10 revolutions. The presence of the "idle" stroke has an adverse effect on the operation of the automatic control.

In the redesigning of small furnaces, the cables of the drive for moving the stand were replaced by a rack. This produced a considerable reduction in the "idle" stroke in the mechanisms for moving the electrode.

In the 20-ton furnace in which a rack drives the telescopic stand, a new reducing gear was fitted in

*The main participants in the work were: from the "Elektrostal' " Plant—M. I. Zuev, V. S. Kultygin, V. S. Laktionov, V. I. Simonov, V. V. Timoshenko, V. P. Tsukanov, B. M. Plevako, I. A. Nazarkin, S. F. Polunin, Ya. V. Gancho, A. M. Artamonov, G. D. Korolev; from TsLA—A. N. Kotikov, V. E. Pirozhnikov, V. V. Stepanenko, B. A. Znamenskii, E. S. Genishta, O. G. Filin.

which, instead of two worm transmissions, there are one worm and two cylindrical gear transmissions. This resulted in a reduction in the "idle" stroke in the system to 0.15-0.35 of a revolution of the motor shaft.

To facilitate the operation of the motor when lifting the electrode, the electrode movement system usually has a counterbalance, the weight of which with the electrode removed should be somewhat less than the weight of the column and the flexible pipe with the electrode holder. This is necessary so that the electrode holder without the electrode can be lowered. However, in this system, if the electrode is in the electrode holder, the unbalanced weight is increased by the weight of the electrode itself and is, for example, in 20-ton furnaces, 2 tons, and in 180-ton furnaces, 6 tons.

Naturally, the greater the unbalance of the system, the greater the power of the motor needed and the poorer the conditions of operation of the automatic controls.

At the redesigned furnace of the "Elektrostal' " Plant an "overblanced"system has been installed in which the weight of the counterbalance is greater than the weight of the descending parts (without electrode). In this case the electrode holder with the electrode removed is lowered by the motor, under the action of the rack on the column of the telescopic stand by means of a spring. When nonconducting material enters the furnace the value of the possible pressure of the rack on the electrode is limited by the final switch, which disconnects the motor if the stress in the spring, determined by reduction in its length, reaches a given value. With the new system it is possible to almost halve the unbalanced weight and the power required in the motor and also to increase the sensitivity of the controls.

New devices for controlling and regulating the process. To select the best electrical and themal systems for furnace operation and controlling the process, the chemical composition and temperature of the metal and slag must be determined during melting. Measurements must be made of the temperature of the inner surface of the lining, the quantity of electrical power supplied to the furnace, the useful power of the arc, the value of the thermal losses and a number of other values characterizing the furnace operation. At the furnaces of the "Elektrostal' " Plant, use is made of certain new devices for controlling and measuring a number of features of the melting process†.

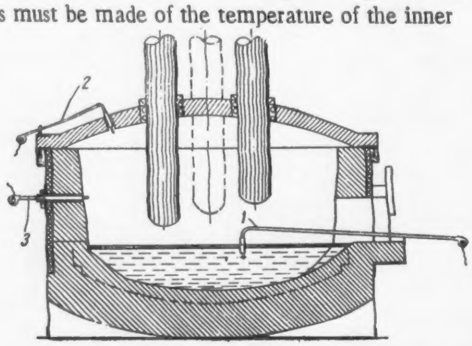


Fig. 1. The position of the thermocouples in the furnace for measuring the temperatures of the molten metal and the inner surface of the lining: 1) immersed thermocouple; 2) roof thermocouple; 3) thermocouple for measuring the temperature of the walls.

The temperature of the inner surface of the lining is measured continuously during the melting in these furnaces.

The device for checking the lining temperature consists of a thermocouple and a secondary recording and indicating instrument mounted at the control panel for the furnace. During the fusion process (after crumbling of the whole charge) the thermocouple is introduced through a special hole in the wall of the furnace (Fig. 1) and is only removed before the metal is tapped. The operator has a schedule for the melt, indicating the required power and the voltage of the arcs to obtain the permissible temperature in the lining.

Checking the temperature of the lining indicated that in many heats, including melting of ShKh15 steel with a fresh charge it is possible to work longer at a higher level of voltage and power without the lining fusing. This leads to a reduction in the length of the melting period.

The treatment of 115 heats of ShKh15 steel showed that during the melting period and the heating of the metal (when working with the remelting method) and also in the oxidation period (for heats with fresh charge) as a rule, the lining was not heated above 1750°C.

Both furnaces have adopted periodic measurement of temperature of metal in the bath (8-10 times during the melting) by new immersed thermocouples consisting of an alloy of tungsten with rhenium. Improved control of the electrical circuit, more careful preparation of the thermocouples and the measurements, together with the use of a new thermal electrode material have reduced the maximum error in measuring the average temperature of the metal from 4.5-5.0 to 2.0-2.5% (for 90% of the cases it did not exceed 1.2%).

New devices have been installed for the continuous determination and recording (by means of two instruments on the control panel for the furnace), during the melting, of heat losses from the cooling water and radiation through the working opening. The heat losses from the water are determined by continuous measurement of the flow and the temperature of the inlet and outlet water‡.

The thermal losses due to radiation, depending on the time the opening is uncovered and the period of

† The new devices were developed with the participation of the Leningrad and Kharkov Refractory Institutes, the Podol'skii Plant for refractory components and the Moscow Electrical Lamp Factory.

‡ For more details see Metallurg., No. 5 (1959). [See English translation].

melting, are also recorded by an instrument. The determination of these losses when determining ShKh15 steel showed up to 70-75% of them occur during the oxidation period of melting. On the whole, the time during which the opening is uncovered comprises 30-40% of the whole melting time.

To control the electrical circuit at both furnaces, special computers have been installed which give an error not exceeding 2.0%. For any short period of melting the instruments can maintain given average values of the currents of the phases or a given quantity of electrical power which should be fed to the furnace during the given melting period.

Investigations of thermal, electrical and metallurgical aspects of melting. The presence of objective measurement in the melting of ShKh15 steel has made it possible to study the thermal and electrical aspects of furnace operation and to establish their connection with the metallurgical processes. In the first place, the investigations were concerned with determining the necessary conditions for carrying out the melting, during which the temperature conditions would be strictly maintained.

The adoption of an objective method for measuring the temperature of the metal (8-10 times during melting in the bath of the furnace and immediately after tapping the metal in the ladle) has established the connection between the temperature of ShKh15 metal during melting and the quality of the finished metal (the quantity and size of the non-metallic inclusions, the state of the ingot surfaces, etc.)

In particular, it was found that to obtain the minimum quantity of oxide inclusions the temperature of the metal at any time during melting, regardless of the period, should not exceed a certain permissible value. As can be seen from Fig. 2, for the conditions of melting ShKh15 steel in furnaces of the "Elektrostal'" Plant, the temperature of the metal during melting should not exceed 1590-1600°C. If the temperature of the metal in the bath at any moment of melting should

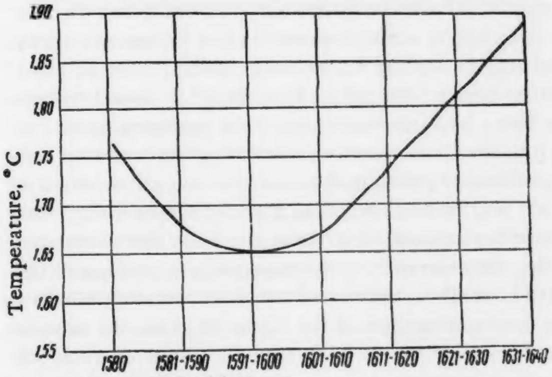


Fig. 2. Graph showing the relationship between the content of oxide inclusions and the maximum temperature of the metal reached during melting.

exceed the above indicated value, the quantity of oxide inclusions in the finished metal increases. The temperature of the metal on tapping is usually 1560 ± 10°C.

As a result of the investigations, an efficient temperature system was developed for melting ball-bearing steel; this is shown in Fig. 3.

The measurements of the temperatures of the metal during the same periods, carried out on a large number of ShKh15 steel heats, showed considerable fluctuation in temperature from heat to heat. Thus, for various heats the values of the temperatures for the same periods differed by 100-110°C and in the ladle by 90°C. The introduction of regular temperature measurement for the metal and control of the electrical system, with the observation of a given temperature system in the metal, led to a considerable reduction in the fluctuations of these temperatures and in more than 55% of the heats it was possible to keep the temperature system during melting with fluctuations not exceeding 20°. At the same time there was a reduction in the content of nonmetallic inclusions and an improvement in the quality of the ball-bearing steel.

The first experiments in using the new instruments for checking melting and for controlling it indicated the possibility of improving the technical and economic features of the furnace operation. Thus, at one furnace, in the melting of ShKh15 steel with fresh charge and using the method remelting**, it was possible to reduce the melting time with current on by an average of 33 and 17 minutes respectively with the specific consumption of electrical power kept at an average of 50-55 kw-hr/ton.

The first work on complex automation of the steel melting in arc furnaces has therefore resulted in an improvement in their operation and has opened up possiblilities for further increase in the standardization of the process, improvement in the quality of the metal melted and an increase in the furnace output.

**According to the results of heats during February-March 1959, with operation of the furnace at 240 volts during the fusion of the metal, in comparison with the period of furnace operation at 234 volts before the redesigning of the electrical equipment and the installation of new devices for measurement and control.

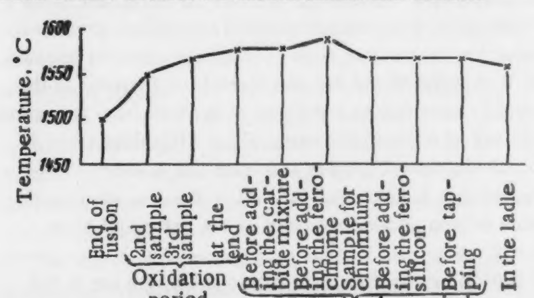


Fig. 3. Temperature system for melting ShKh15 steel.

THE DESIRABILITY OF USING SCRAP IN CONVERTER PRODUCTION

S. G. Afanas'ev and B. E. Gurevich

Central Research Institute for Ferrous Metallurgy

In the production of steel in converters by top blowing the iron with oxygen, a much greater quantity of heat is evolved than is needed to heat the bath up to the required temperature. In practice, this excess heat is used for melting the steel scrap or reducing the iron ore. The total number of coolers needed to obtain a normal final temperature in the metal, based on the scrap, reaches 20-30% of the weight of iron, depending on the temperature and chemical composition of the iron and the amount of heat losses in the converter.

The selection of the cooler for converter melting is very important since it largely determines the organization of work in the shop, the running of the process and the technical and economic operating factors.

The use of steel scrap as the cooler has considerable advantages: The scrap does not bring a large quantity of silica into the bath, needing additional lime for slagging and having an unfavorable effect on the durability of the lining; the iron also gives a constant cooling effect. However, when cooling with scrap it is essential to have special charging methods and to spend time on the charging which lowers the rate of production of the converter. Thus, according to the calculations of Gipromez, with a total cycle for a 75-ton converter smelting of 40 minutes, the time for charging the scrap is 2.5 minutes, or 6.25% of the total time. Furthermore, when charging heavy return scrap there is extra unavoidable mechanical wear of the lining at the points where the scrap falls against it.

The use of iron ore as the coolant, when all the metallic part of the charge consists of molten iron, has its advantages; there is an increase in the productivity of the units, their maintenance is simplified, there is a reduction in the consumption of gaseous oxygen because of the oxygen in the ore and there is an increase in the yield of steel due to reduction of the iron from the oxides. One ton of reduced iron costs about 110 rubles when 1 ton of ore costs 50 rubles and when the amount of extracted iron is about 80% of that in the ore, whereas the price of iron under conditions at the Petrovskii Plant is 328 rubles/ton, and the scrap (depending on the grade) is 160-250 rubles/ton. In existing shops, the ore is fed into the converter mechanically; the time for feeding the ore and lime does not exceed 40-60 seconds; the free-flowing material can be added during blowing; at this moment it is merely necessary to somewhat reduce the intensity of oxygen addition to prevent the removal of a large quantity of fine particles. As a coolant, however, ore has some serious drawbacks. These include the high content of silica in it (up to 15%), the varying quantity of iron oxides and also the different degree of assimilation of the oxides, depending on the conditions of charging and the lumpiness of the ore.

At the present time there are two approaches to the problem of selecting the type of coolant for existing and for newly built converter shops. In the conversion of open-hearth iron by top blowing outside the USSR, mainly steel scrap is used; in existing Soviet plants the blast is cooled with iron ore. However, the use of iron ore in Soviet converters is to a large extent unavoidable and is mainly due to the fact that plants working with top blowing are the result of the redesigning of Bessemer plants and do not have charging facilities for adding the scrap.

The advantageous use of steel scrap in foreign converter plants is connected with economic considerations; the cost of scrap is usually less than that of iron and in this sense the use of scrap provides a lower cost for the charge and consequently for steel.

However, scrap is a valuable substitute for iron in the charge of steel smelting units, therefore the difference in prices of scrap and iron is arbitrary to a certain extent. Under conditions of a planned national economy, taking into account the economic effectiveness of a given process, the cost of these materials should be taken as being the same, as is done at the present time in Gipromez in comparative calculations for converter and open-hearth plants. In this case, the cost of the converter steel will be much less than that of the open-hearth steel since the expenses in conversion—a basic factor characterizing the economics of the process—are much lower in the first case. Thus, in the far-from-ideal operating conditions in the converter shop of the Petrovskii Plant, the expenses on conversion per ton of steel in the converter are 30-35 rubles less than in the open-hearth furnace. The higher cost of converter steel (by 40-70 rubles) is mainly due to the higher cost of the metallic part of the charge—by 100 rubles or even more.

The fields of application of scrap should be determined from considerations of the best organization of transport, mechanization of the charge, technological advantages and balance. From this point of view the use of scrap in open-hearth production is undoubtedly more desirable since the open-hearth furnace is the most convenient unit for resmelting scrap. In the converter shops the use of scrap leads to a reduction of the productivity of the unit; in the open-hearth furnaces, on the other hand, the productivity is only reduced for an exceptionally high (more than 60-65%) content of iron in the charge. The reduction of output due to the exceptionally high content of iron in the open-hearth charge causes a corresponding increase in the cost of steel. There is naturally a reduction in the life of the furnaces due to the considerably longer polishing period.

Thus, in the charge of open-hearth furnaces it is advisable to use scrap and in the converters it is best to use 100% molten iron and to cool the bath with iron ore, which not only increases the yield of useful metal, but also introduces a large amount of oxygen into the bath. The amount of combined oxygen introduced by the iron ore when it is used to the extent of 8% of the weight of the iron, with a Fe_2O_3 content of 85% and a degree of assimilation of the order of 80%, is 11.5 m^3/ton iron; there is a corresponding reduction in the consumption of gaseous oxygen of high purity. Furthermore, the addition to the converter of a large amount of iron oxide has a favorable effect on slag formation which in this case proceeds more intensely.

However, the addition to the converter of iron ore having up to 15% and even more SiO_2 sharply increases the amount of slag in the converter, which has an unfavorable effect on the durability of the basic refractories, the wear of which is determined to a large extent by the action of silica at high temperatures. This is a basic drawback in the use of ore as the coolant. It is not known, however, to what extent the durability of the lining improves if iron ore is not used and the bath is cooled with scrap; there are no data on the comparison of these coolants and this interferes with the more accurate quantitative determination of the degree of "harmfulness" of the ore with respect to the lining. It should be borne in mind that the scheme of operation used in our operating and planned shops in which, as a rule, for two operating converters there is one reserve converter, even with the present durability of the refractories, provides steady operation almost without loss of production. The durability of the refractories therefore has little effect on the productivity of the converter shops. On the other hand, the cost of the converter brick in the cost of the steel, for example at the Krivoi Rog plant, is 17-19 rubles per ton; assuming in the case where ore is not used that the consumption of refractories per 1 ton of steel is reduced by 30%, then the saving in materials amounts to about 6 rubles/ton, whereas the productivity of the converter due to the longer smelting, as shown previously, is reduced by 6.25%. Furthermore, the fact that from the cheap iron ore in the converter, neglecting the blast-furnace smelting, valuable steel is obtained, for the production of which no iron or scrap is expended, leads to an uneconomic effect due to the increase in durability when operating with scrap.

It follows from the above that even with the comparatively low durability of the refractories, iron ore should be used as the coolant in the converter bath and scrap should be sent to the open-hearth furnaces.

Of course, this should not be taken too dogmatically: In a number of cases the use of steel scrap in the converter production can be more desirable. This is true of those types of converter production where a high concentration of a certain oxide is required in the slag, e.g. the conversion of iron with increased content of phosphorus (Kerchen and Lisakov type) with the production of phosphate slag, and the conversion of high-manganese iron of the Kremnikov deposit (Bulgaria), where the primary slag with high content of MnO is converted into ferromanganese or silicomanganese. In these cases the slag must not be diluted with large quantities of silica from the ore.

Furthermore, it is sometimes undesirable to send to the consumer the scrap existing at the plant. If, for example, it is necessary to transport it large distances, we may better process it on the spot in the converters.

In selecting the coolant, therefore, special attention should be given to each individual case, using the actual conditions as a basis.

However, even when cooling the bath with scrap it is desirable to have more accurate control of the temperature of the smelting, adding iron ore in small amounts since the total mechanization in the addition of loose ore makes it possible to add it in any quantity and at any moment without stopping the blowing.

Furthermore, when cooling the bath with iron ore, if possible the harmful effect of silica and the large quantity of iron should be reduced. For this purpose it is necessary to add a large part of the ore with the first portion of the loose material and after 5-6 minutes of blowing to tap the intermediate slag, thereby removing a large amount of silica from the converter at a fairly low bath temperature. It therefore follows that the iron ore used in converter production should contain a minimum amount (not more than 10%) of silica; the lime should not contain a large amount of incompletely burnt material and overburnt material, etc. Since the lime is of high quality, the slag can be given the required basicity and the harmful effect of SiO_2 on the lining can be reduced.

In the cooling of the bath, good possibilities are offered by pellets made from concentrates of iron ores containing about 70% iron and 2-5% SiO_2. With a

good cooling effect, low cost and the advantage of being easily transported and charged into the converter, the pellets have the favorable qualities of ore andscrap, but do not have their disadvantages.

The use of iron ore or pellets in converters will definitely improve the scrap balance and will mean that the open-hearth furnaces can work with a small amountof iron in the charge. Bearing in mind that the Seven-year Plan for the development of the national economy stipulates the smelting of more than 9 million tons of steel in converters with a top oxygen blast by 1965, the improvement in the scrap balance can be very important.

* * *

EXPERIMENTAL CONTINUOUS MEASUREMENT OF THE TEMPERATURE OF MOLTEN STEEL IN THE LADLE AND FURNACE

P. Ya. Sorokin

Institute of Metallurgy of the Ural Branch of the Academy of Sciencesof the USSR

The coefficient of distribution of the elements between slag and metal and also the solubility of gases in the metal, which to a large extent influence its quality, are in direct relation to the temperature of the metal. A knowledge of the true temperature of the molten steel in the furnace during the course of the melt and particularly in the working period is the principal prerequisite for the correct conduct of the melting process and for the production of quality steel.

For correct pouring, it is also very important to know the true temperature of the metal in the ladel, especially in the lower layers.

Until quite recently, the temperature of the metal in the steel-melting process was determined visually. During the development of the practice, the temperature of the steel was determined first by means of optical pyrometers and then by immersion thermocouples. Such an occasional character of the temperature measurements, however, is unable to satisfy the requirements of steel workers. There is a real requirement for continuous measurement of the temperature of the molten steel in the furnace during the working period, and in the ladle during pouring. The caps of fused quartz and alundum, currently used for the protection of the hot junctions of thermocouples, however, do not satisfy the requirements. Quartz caps soften in molten steel and allow hardly more than one temperature measurement to be made. Alundum caps, although more refractory, are less reliable than quartz caps, since they crack when immersed in molten steel.

In 1952–1955, the Institute of Metallurgy of the Ural Branch of the Academy of Sciences of the USSR developed a technique of making thermocouple caps based on zirconium dioxide, possessing considerable heat resistance and high refractoriness. These caps satisfy the principal requirements for the continuous measurement of the temperature of molten steel: They do not crack when immersed in the metal without previous heating. Their thermal inertia does not exceed 60-70 sec for a wall thickness of 3 mm. This time of heat transmission of the protective caps does not constitute an obstacle to the continuous measurement of molten steel during the melting process and during pouring, both of which take a much longer time.

The laboratory testing of these thermocouple caps was carried out in a high-frequency furnace. In the first experiments, thermocouples protected by caps of zirconium dioxide were inserted through the bottom into magnesite crucibles, in which steel was melted and kept in that condition. After each melt, the magnesite crucible was cut or broken open for inspection of the caps. It was found that the caps were not destroyed after they had remained in molten steel for two hours at a temperature of up to 1780°C. In other experiments, thermocouples, protected by caps made on the basis of zirconium dioxide, were inserted in the bottom of a high-frequency furnace in which metal was melted. The total duration of satisfactory functioning of the thermocouple in the continuous measurement of the temperature of molten steel in four melts was 15 hours.

The experiments on the continuous measurement of the temperature of molten steel in the ladle were carried out in cooperation with workers of the Ural Railroad-Car Building Factory, one of the machine-constructing factories and the A. K. Serov Metallurgical Combine in Serov.

The hot junction of assembled thermocouples was introduced into the ladle through a false stopper at the bottom or side of the ladle and was located at a distance

of 200–300 mm from the ladle bottom. These measurements showed that the temperature of the upper layers of the metal in the ladle was higher than the temperature of its lower layers. The drop in temperature between the upper and lower layers in 30–45 ton ladles was 100–125°C and as an average of 15 melts in a 45 ton ladle it was 77°C.

The experiments showed that with increase in the temperature of the metal in the ladle toward the end of pouring, measured by a continuously operating thermocouple, the temperature of the steel, determined by means of an optical pyrometer, falls regularly, in agreement with our usual concepts.

The experiments on the continuous measurement of the temperature of molten steel during the course of the melt were made in an acid open-hearth furnace of the A. K. Serov Metallurgical Combine. The thermocouple was mounted on a watercooled tube, protected from direct contact with the metal by ordinary fireclay stopper sleeves, not specially designed for the purpose. The over-all weight of the assembled thermocouple therefore amounted to 150 kg. The external appearance of the thermocouple after continuous measurement of the temperature of the metal in the furnace for 70 min is shown in Fig. 1.

The thermocouples were introduced into the furnace through the wicket of a charging door. The hot junctions were immersed in the metal to a depth of 100–125 mm. Figure 2 shows the metal temperature curves in the working period of two melts, measured by continuously operating thermocouples, the hot junctions of which were protected by caps made in the Institute of Metallurgy of the Ural Branch of the Academy of Sciences. In Fig. 2, the crosses indicate the temperature of the metal as measured by an ordinary tungsten-molybdenum immersion thermocouple, the electrodes of which were taken from the same coils of wire from which was assembled the experimental, continuously operating thermocouple. The experiment showed that in one case, the temperatures measured by the continuously operating thermocouple and by the ordinary immersion thermocouple were in agreement, while in the other case, they differed by 10-12°C. It should be noted that the continuously operating thermocouple was not protected by an inert gaseous medium. This experiment permits one to consider that the variation of the thermoelectromotive force of electrodes of tungsten and molybdenum wire caused by their remaining in the molten metal for 40 min (or even longer) cannot form an obstacle to the continuous measurement of the temperatures of molten steel. These curves show that the thermocouple detects with sufficient rapidity the drop in temperature of the bath caused by the addition of considerable amounts of solid ferrochrome to the metal.

Figure 3 shows two caps after remaining in the metal bath of the furnace for 65 and 88 min. As will be seen from the figure, the fireclay plugs surrounding the caps and in which they were mounted have been eroded on the entire surface to a depth of 5–6 mm, while the caps, made on a zirconium dioxide basis, were fully preserved.

Since the first experiments showed that the fireclay protective surrounding was little eroded in an acid open-hearth furnace, the thick fireclay stopper sleeves were replaced in subsequent experiments by lighter ones, viz., bottom-pouring sleeves. The condition of a cap, after the thermocouple, protected by the lighter sleeves, had been in the furnace for 55 min, is shown in Fig. 4b. For comparison, Fig. 4a shows the cap before the test.

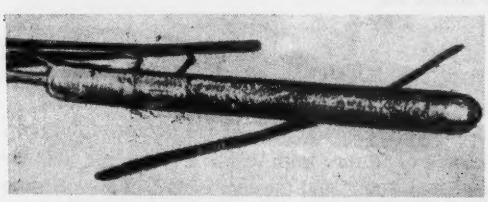


Fig. 1. Thermocouple after continuous measurement of the temperature of the metal in an acid open-hearth furnace.

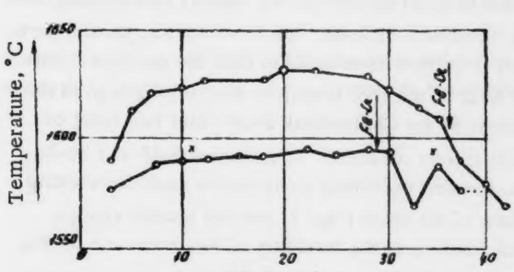


Fig. 2. Curves of temperatures of the metal measured in the furnace in two melts by a continuously operating thermocouple.

Fig. 3. Condition of zirconium dioxide caps after continuous measurement of the temperature of the metal in an open-hearth furnace for periods of 65 and 88 min.

The experiments made at the A. K. Serov Combine* demonstrate the possibility of organizing the continuous measurement of the temperature of molten steel in an acid furnace by means of thermocouples, the junctions of which are protected by caps made on the basis of zirconium dioxide, while the watercooled tube is insulated by fireclay refractory sleeves. For continuous measurements of the temperature of molten steel in basic furnaces, insulating armored sleeves using a basic refractory material will be necessary.

For the introduction of continuous measurement of the temperature of molten steel in furnaces and ladles, organizations producing highly refractory materials should be instructed to carry out the mass production of thermocouple caps on the basis of zirconium dioxide, after having utilized and perfected the technology of their production as developed at the Institute of Metallurgy of the Ural Branch of the Academy of Sciences.

Continuous temperature measurements will greatly assist the introduction of automation in steel-melting production according to the chief parameter of the entire process—the temperature of the metal.

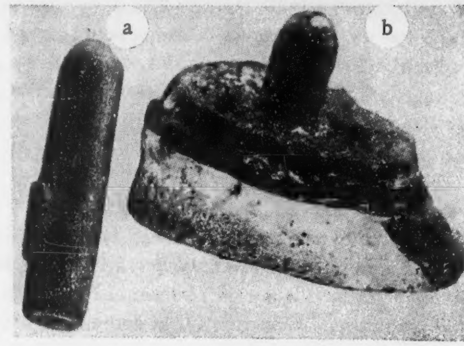


Fig. 4. Condition of thermocouple caps made of zirconium dioxide; a) before testing; b) after continuous measurement of the temperature of molten steel in the furnace for 55 min.

* The following took part in this work: V. F. Isupov, P. P. Semenenko, V. G. Tyulebaev, V. A. Nosov, A. A. Chepurnova, G. A. Bolotov, A. S. Gorbunov, E. I. Mosyagin.

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THE PERFORMANCE OF ROOFS OF VARIOUS THICKNESSES IN ELECTRICAL STEEL-MELTING FURNACES

M. N. Kaibicheva, N. A. Tulin, and N. I. Fadeeva

Eastern Refractories Institute and Chelyabinsk Steel Plant

The linings of electrical furnaces work under conditions of high concentration of melting dust, variable gas medium and high, sharply fluctuating temperatures in the working space.

Tests have shown that the concentration of melting dust in the space below the roof of an electrical furnace, depending on the period of melting and the grade of steel being melted, varies from 0.76 to 50.5 g/m^3, reaching a maximum during the blowing of the bath with oxygen. The melting dust contains 74% of iron oxide and an increased amount of manganese and magnesium oxides. This composition characterizes its low refractoriness (1480-1560°), which in a reducing medium is reduced by 250-270°C. The melting dust, saturating the refractory materials, leads to physicochemical and thermal lack of uniformity in the various zones of the lining and also to a reduction in the refractoriness of its working zone.

The development by metallurgists of a melting system with low evolution of dust with a constant gaseous medium (preferably oxidizing) would considerably help to lengthen the life of the lining in electrical furnaces.

The lining of an electrical furnace works under severe temperature conditions. At the center of the roof, at a distance of 50 mm from its hot surface, the fluctuations reach 300-600°C and the rate of fall in temperature is 150°C per minute. Along the periphery of the roof the temperature fluctuations are much less.

The difference in the temperature gradient through the thickness of the roof at the central and peripheral parts leads to a different rate of wear. The central part wears 2-4 times more quickly than the peripheral part. The wear of the roof is mainly due to splitting. At the furnaces of the Chelyabinsk Steel Plant two types of splitting were observed: of thickness 5-15 and 30-80 mm, the first occurring along cracks near the working surface of the brick (Fig. 1) and the second along a crack forming at the boundary of the dense and friable zones (Fig. 2). The wear of the central part of the roof is mainly due to large splits. The splitting and consequently the wear in the central part can be reduced by reducing the temperature gradient across the crack of the roof at a given section.

This can be achieved by increasing the thickness of the quicker wearing central part.

On this basis, a design of a differential thickness roof was developed and tested in service with electrical furnaces.

The tests were carried out on arc furnaces. The thickness of the peripheral part was (as usual) 230 mm; the central part, 300 mm (MKhS 2 and 9). The diameter of the thickened part was 1950 mm with a roof diameter of 3160 mm (small furnace). The roof was laid down in an annular manner. The laying was carried out in the usual pattern with the peripheral part filled with sand.

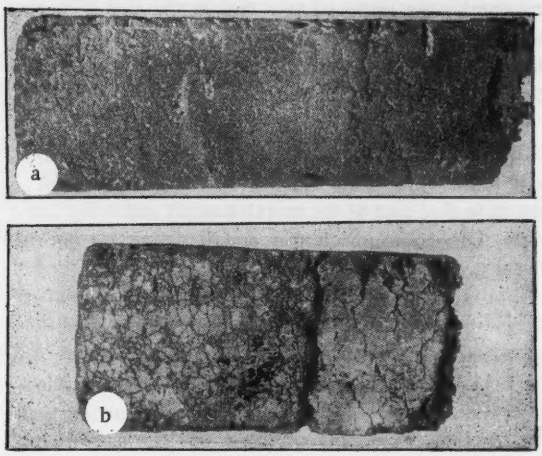

Fig. 1. Character of cracks of roof brick: a) near the working zone; b) at the boundary of the dense and friable zones.

There are three ways for laying a roof of differential thickness:

1) the thickened part only extends upwards;
2) the thickened part extends equally both up and down (Fig. 2);
3) the thickened part only extends to the working case of the furnace.

When laying the roof according to the 1st and 3rd methods, the transition from the ordinary to the thickened part of the roof should be gradual. This is achieved by using wedge-shaped bricks of the same length for the thickened part of the roof. In the case under consideration, the roofs were laid according to the second variant. The steels melted in furnaces with differential thickness roofs were carbon, constructional, ball-bearing, high-speed, tool, heat resistant, chrome-tungsten and other steels.

The use of differential thickness roofs in arc furnaces has extended their life from 90-100 to 160-185 melts. At the present time the roofs of these furnaces with 300 mm brick are laid exclusively in different thicknesses. The achieved life of the roofs is by no means a limiting figure. The roof is not worked to the end due to the appearance of holes in the economizers, during the repair of which, due to the annular laying, the central part is destroyed at the place where the electrodes decompose.

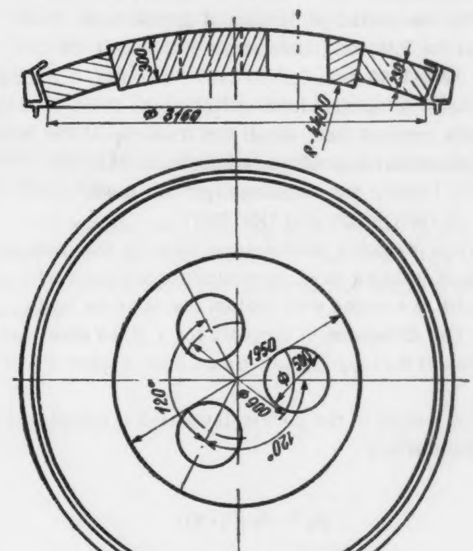

Fig. 2. Design of a different thickness roof of an electrical furnace.

The use of a differential thickness roof with efficient laying and strengthened electrode openings with concrete protection of the economizers will lead to the elimination of intermediate repairs connected with the replacement of the central section of the roof; it will also reduce labor costs, the consumption of materials in repairs and considerably extend the life of the roof.

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THE PRODUCTION OF DOUBLE-LAYER SHEET STEEL

M. I. Bazhenov-Korchemnyi and P. N. Mazurik

Deputy Mill Manager and Section Manager
The Sheet Rolling Mill of the Kuznets Steel Combine

On the suggestion of G. V. Sharov and G. I. Goncharov, members of the Planning Department, our combine has started production of double-layer sheet steel of the following dimensions: thickness 5-50 mm, width 1000-1500 mm, length 2000-7000 mm, cladding layer 0.5-5 mm, maximum weight of the sheet 1200 kg. The base layer of these sheets was made up of the following commercial and alloyed steels: St.3, 15k, 20k, SKhL45, 12MKh; the cladding layer was made up of steels ÉI 496(OKh13) and 1Kh18N9T.

When preparing double-layer steel by this method, a plate of stainless steel, unsymmetrically placed in the mold, is covered with molten metal of the basic layer. The dimension of the plate for a given sheet and thickness of the layer are calculated from a special formula.

The weight of the double-layer slab is calculated from the formula

$$p_2 = k_2 \cdot p_1 \cdot n_1,$$

where p_1 is the weight of a given sheet; n_1 is the number of sheets obtained from one rolling; k_2 is the fabrication factor of the slab-sheet. For steel 1Kh18N9T with weight of sheet 900-1000 kg, k_2=2.1, for the steel OKh13 and with a weight 1100 kg, k_2=1.7.

The ratio of the weights of ingot and slabs obtained from this ingot are determined from the equality

$$p_3 = k_3 \cdot p_2 \cdot n_2,$$

where p_2 is the weight of the double-layer slab; n_2 is the number of slabs obtained from one ingot; k_3 is the fabrication coefficient of the ingot-slab. For the cladding layer of steel 1Kh18N9T, k_3=1.65; for OKh13 steel, 1.45.

After rolling, both surfaces of the plate are specially machined and the plates are suspended on soft wire of diameter 4 mm in molds which are covered with sheet iron.

The base metal is melted in the open-hearth furnace. To obtain double-layer ingots, the steel is bottom-poured into molds with iron hot tops. The gases evolved during the pouring are removed by a fan. At the end of pouring, an iron cramp is placed in the top part of the ingot and the melt is soaked on the platform for 1-1½ hours and then sent to the blooming mill.

The stresses forming in the basic metal due to different shrinkage between it and the plate sometimes lead to the appearance of transverse and longitudinal cracks in that part of the metal directly adjoining the plate. The cracks which form are welded during hot rolling at the blooming and sheet mill.

In the soaking pits the ingot is arranged so that its side with the covered plate is turned in a direction opposite to the wall of the compartment (determined from the cramp in the top part of the ingot). As a rule, the ingots enter the pits with a temperature of not less than 700°C. If the temperature is below 600°C, the compartments are heated to 900°C. The ingots are heated to 1310-1320°C, then soaked for 1½ hours at 1300-1310°C and sent to the blooming mill with the covered plate upwards.

The double-layer ingots are rolled with the maximum permissible reductions. (If necessary, turning is decided after preliminary total reduction of not less than 50%). In removing the head and tail parts of the rolled material, not less than 100 mm is cut off the front and back end of the plate.

A certain unevenness in the thickness of the cladding layer depends on the plasticity of the metal of both layers, determined by their chemical composition: The greater the difference in plasticity the more uneven the thickness of the layer. For example, when using St. 3 steel with 1Kh18N9T steel, the unevenness in the thickness of the layer is ± 8%; and when using ÉI496 steel with St. 3 steel, from +25 to -19% of the average thickness of the layer. This unevenness is preserved with further rolling of the slab to sheet.

The mechanical properties of a double-layer sheet are similar to those of the steel of the base layer. The resistance to shear between the basic and the cladding layers is 25-40 kg/mm^2.

Before being placed in the furnaces of the sheet rolling mill, the double-layer slabs are cooled in stacks, protected from drafts and then arranged for flame trimming.

Sometimes during rolling at the blooming mill, the layer of the soft metal tears and sharp edges are formed

from the metal of the plate; they must be smoothed off by a pneumatic chisel otherwise blisters may form during further rolling at the sheet mill. The rolling of slabs with a plate of 1Kh18N9T steel is carried out with the plate at the bottom, and from OKh13 steel with the plate at the top. This is because at a rolling temperature of 1300-900°C, the plasticity of a layer of steel OKh13 is greater than the plasticity of the base layer, and the plasticity of 1Kh18N9T steel is less than that of the base layer and during rolling with the cladding layer on top, the sheet "binds" the top roll.

The heating is carried out as for St. 3 steel and the rolling as for 1Kh18N9T steel or also according to St. 3 steel. As a rule, the slab is rolled across its main axis. The reduction for the first longitudinal pass was equal to the thickness of the material poured to smooth out the side ridges, limiting the plate and the layer; the slab was then rotated through 90° in a horizontal plane and then rolled transversely.

In the rolling of steel with a cladding plate of 1Kh18N9T steel, the scale is removed by a hydraulic apparatus using rods; and in the case of OKh13 steel plate, on a grooved roll in the roughing stand without the addition of rods.

During cooling, the sheets buckle (up to 300 mm) thereby introducing difficulties into their transport and machining. To prevent this, a special bending machine must be installed in the line.

Sheets with a layer of 1Kh18N9T steel are heat treated at 900-920°C, which removes the work hardening obtained in the last passes in the stainless layer, and somewhat improves the structure of the base layer. Sheets with a cladding layer of OKh13 steel are subjected to high tempering. In those cases where the thickness of the sheets is greater than 28 mm, normalization with high tempering is employed.

After heat treatment, the sheets are dressed and cut to the required dimensions. Sheets with a cladding layer of 1Kh18N9T steel are also subjected to pickling. The difficulty of this process consists of the fact that to pickle the layer of 1Kh18N9T steel it is essential to have an aggressive medium which attacks the basic layer. A method has been developed at the Kuznets Steel Combine in which the stainless layer is covered with a special paste. After 12-48 hours soaking, the sheets are etched in the usual solution for stainless sheets for 10-15 min.

This method for producing double-layer steel has not yet found wide application. It can only partially satisfy the requirements of the chemical industry for this steel.

* * *

ON AN EFFICIENT LAYOUT FOR BLOOMING MILLS

L. V. Andreyuk

Calibrator of the Magnitogorsk Metallurgical Combine

Great work has been performed at the Magnitogorsk Metallurgical Combine to increase the productivity of two blooming mills. Only within the last five years the productivity by input of blooming mill No. 2 has been increased 20% and of blooming mill No. 3, by 42%. The most important factor in the increase of productivity during this period was the installation of two additional nonreversing stands at blooming mill No. 2 and a 1100 reversing stand at blooming mill No. 3. The present arrangement of blooming mill equipment is shown in Figs. 1 and 2. Another two additional stands which were part of the 630 continuous billet mill had been installed somewhat earlier at blooming mill No. 2.

Thus the number of rolling stands operating in a single flow with blooming mill No. 2 is 17. Blooming mill No. 3 was converted to a double blooming mill.

The presence at one and the same plant of two blooming mills, whose power was increased by various methods, makes it possible to compare two ways of increasing the productivity of blooming mills.

Table 1 shows the approximate data on the expenditures for installation of additional stands at both blooming mills.

The productivity of blooming mill No. 2 by input in 1954 as compared with 1952 (the additional stands were installed in 1953) increased by 319,362 tons/year and the capital expenditures per ton of increase of the annual productivity resulting from the data in Table 1 were

$$\frac{3,777,700}{319,362} = 11.83 \text{ rubles}$$

The productivity of blooming mill No. 3 in 1956

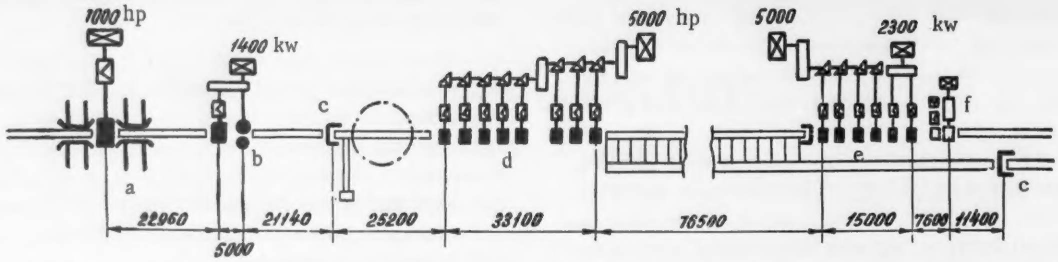


Fig. 1. Layout of the equipment at blooming mill No. 2: a) 1150 reversing stand; b) additional nonreversing stands; c) shears; d) 630 continuous billet mill; e) 450 continuous billet mill; f) electrical planetary flying shears.

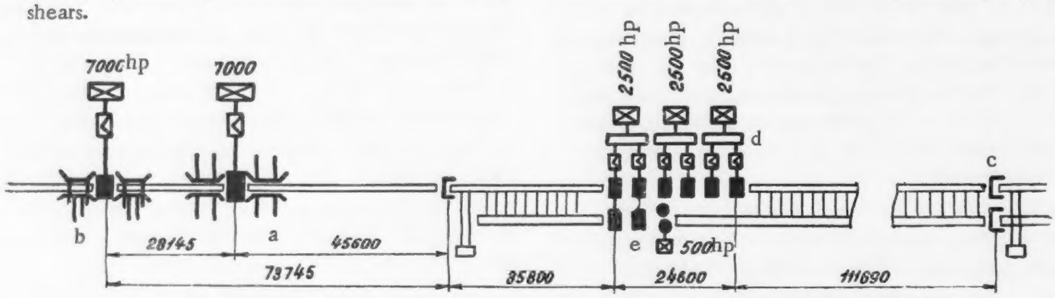


Fig. 2. Layout of equipment at blooming mill No. 3; a) 1150 reversing stand; b) additional 1100 reversing stand; c) shears; d) 720 continuous billet mill; e) group of slabbing stands..

as compared with 1954 (the 1100 stand was installed in 1955) increased and the capital expenditures per ton of annual productivity were 15.98 rubles.

For comparison it is possible to cite the capital expenditures per ton of annual productivity for the slabbing mill we recently put into operation; these were about 68 rubles. This magnitude for the redesigned blooming mill, according to the data of Giprostal', is 51.5 rubles.

It can be seen from the cited data that the increase in the productivity of the blooming mills by increasing the blooming facilities with a more complete utilization of the soaking pits, of the shears for hot cutting, of the storage areas, etc., is attained with considerably less specific expenditure than would be required for building a new blooming mill.

The initial capital expenditures per unit increase of productivity increased with the installation of the second reversing stand. However, preference should nevertheless be given to this variant in comparison with the installation of additional nonreversing stands. Its advantage lies in the fact that with the specific initial expenditures the blooming mills with two reversing stands can in the future continue, without additional expenditures, to increase the productivity by an increase in the heating facilities, by an increase in the output capacity of the rear section, etc. Such possibilities are very limited for a blooming mill with additional nonreversing stands.

It follows from Table 2 that the increase in the annual productivity of blooming mill No. 2 occurred parallel to the increase in the hourly productivity, which is determined mainly by the productivity of the rolling mills. Thus the potentials of the blooming facilities are utilized almost completely. Idle time due to the lack of heated metal were negligible.

Somewhat later, after the installation of the additional stands there was a reconstruction of the electrical

TABLE 1. Capital Expenditures for Installation of Additional Stands at the Blooming Mills of the Magnitogorsk Metallurgical Combine (in prices and norms quoted from July 1, 1950).

Item of Expenditures	Total of expenditures			
	blooming mill No. 2		blooming mill No. 3	
	Thousands of rubles	%	Thousands of rubles	%
Mechanical equipment (with installation)	2649.9	70.1	4017.5	36.6
Electrical equipment (with installation)	483.1	12.8	3826.1	34.8
Construction work and sanitary engineering	260.1	6.9	2329.5	21.2
Electric bridge crane (with installation)	–	–	190.2	1.7
Other expenditures	384.6	10.2	620.9	5.7
Total	3777.7	100.0	10984.2	100.0

TABLE 2. The Increase in the Productivity of the Blooming Mills of the Magnitogorsk Metallurgical Combine after Installation of Additional Stands.

Blooming mill	Indices	Year prior to reconstruction	Years after reconstruction (without consideration of the year when reconstruction was being done)				
			1	2	3	4	5
№ 2	Annual productivity by input, %	100.0	113.6	124.5	125.3	125.5	126.8
	Productivity at the busy hour, %	100.0	115.0	123.8	126.8	127.1	127.5
	Idle times due to lack of hot metal, %	2.8	4.0	2.3	2.4	4.5	2.7
№ 3	Annual productivity by input, %	111.9	141.5	144.1	151.2	156.3*	—
	Productivity at the busy hour, %	111.2	160.1	164.1	167.2	170.9*	—
	Idle times due to lack of hot metal, %	2.5	11.2	11.8	8.1	5.5*	—

* For the first quarter of 1959.

equipment of the 1150 reversing stand, which assured an additional certain increase in the productivity; after this, during the last three years, the productivity of blooming mill No. 2 has increased negligibly.

On blooming mill No. 3 the rapid increase in the productivity still has not kept up with the increase in the hourly productivity (Table 2). This means that the potentials of the reversing blooming stands are still not totally utilized. Idle time due to the lack of heated metal after installation of the second reversing stand increased more than four times. By putting into operation additional heating facilities this idle time was reduced, and the annual productivity increased without any additional expenditures for the group of blooming stands.*

However, the reversing stands in the composition of the blooming mills have great shortcomings.

Thus on the 1100-1150 blooming mills with a standard drive of 7000 hp the working torques are within 180-200 meter-tons. With an acceleration of the motor of 40-45 revolutions per minute per second, and with the magnitude of the flywheel moment of the motor with the mill about 510 meter[2] tons, the dynamic moment of the acceleration is about 30% of the torque being developed by the motor. Consequently, 100 kw of the power of the reversing motor are equivalent to 70 kw of the power of the nonreversing motor. If we take into consideration the necessity of supplying the reversing motor from the transformer unit which approximately triples the power of the electrical machines of the main drive, then the power of the drive of the reversing mill should be greater than the power of the ac motor of the nonreversing mill by $\frac{100 \times 3}{70} = 4.3$ times.

The heavy and high-speed auxiliary equipment is also driven from the dc variable motors. The drive power of the auxiliary equipment of the reversing stands is close or even exceeds the power of the main-drive motors.

*It is expedient to make an estimate of the variants of reconstructing blooming mills starting from the specific expenditures per ton of increase of the planned output and not from the productivity actually attained. Editor.

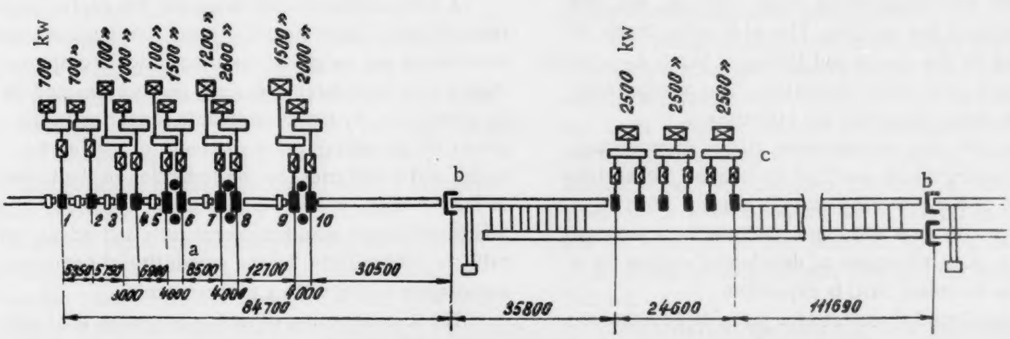


Fig. 3. Diagram of the arrangement of the equipment of a continuous-type blooming mill: a) 1100 continuous blooming mill; b) shears; c) continuous billet mill; 1-10) numbers of stands.

TABLE 3. Regimes of Rolling on a Continuous Blooming Mill

Blooms from ingots with dimendions $\frac{690 \times 620}{750 \times 680}$ mm and a weight of 7.0 tons.					Slabs from ingots with dimensions $\frac{1060 \times 550}{1100 \times 590}$ mm and a weight of 8.6 tons.				
Stand	h, mm	b, mm	Δh mm	n RPM	Stand	h, mm	b, mm	Δh mm	n RPM
1K*	610	630 / 690	80 / 140	3.00	1K	1000	560 / 600	60 / 100	3.00
2K	550	625	80 / 140	3.00	2	520	1005	40 / 80	3.00
3	545	565	80	2.95	3	470	1015	50	3.16
4K	405	590	140	3.80	4	385	1030	85	3.80
5	515	420	75	3.00	5	345	1040	40	2.87
6K	375	445	140	3.80	6	265	1025	80	3.80
7	370	390	75	4.47	7	225	1035	40	4.58
8K	225	415	145	6.75	8	155	1020	70	6.75
9	340	235	75	7.87	9	130	1030	25	7.42
10	250	250	90	9.85	10	100	1010	30	9.85

*K=turning

The total power of the electrical machines of the reversing stand exceeds the power of the rolling mill motor by 3.6-4.9 times. If here we consider only a 70% use of the rolling mill motor then on the reversing stand the power of the electrical machines is more than 5.1-7.0 times that which would be required on a nonreversing rolling mill (it is possible to disregard the power of the slow-speed screw-down devices and the short invariable tables on the nonreversing stand). This means that the power of the electrical equipment of one reversing stand would be sufficient for 5-7 nonreversing stands operating with the same reductions and speeds (although such high speeds on a nonreversing stand are not necessary).

The mechanical equipment of the reversing stand is also heavy, cumbersome and expensive. The great length of the roll bodies increases the dimensions of the stand in width. The powerful and high-speed screw-down mechanism with the large balancing device increases the dimensions of the stand in height, and with it the height of the building. The wide tables should be calculated for the shocks and falling of ingots and,operate under heavy, high-speed conditions. The manipulators should be strong, massive, and high-speed.

If we take into consideration all the shortcomings of the reversing stands and that the needed productivity of certain of our blooming mills is already 3.5-4 million tons per year and more, then the idea proposed by Prof. A. A. Aleksandrov of developing designs for a continuous blooming mill is expedient.

We developed a variant of a group of 10 nonreversing blooming mill stands instead of two reversing stands applicable to the conditions of blooming mill No. 3 (Fig. 3). According to this variant it is possible to roll, on one set of rolls,blooms with a cross section of 250 x 150 and 290 x 290 mm, and also slabs with a maximum width of 1000 mm and a minimum thickness of 100 mm (the entire assortment of blooming mill No. 3 is thus rolled).

All the stands of this blooming mill except for the first and second are arranged in pairs. Each pair of stands operates as a continuous group. When selecting the reduction on the first stand of each pair, starting from the certain, natural bite,there is the possibility of considerably increasing the reduction on the second stands by realizing a forced undertaking of the strip by means of the first stand. The regimes for reducing the blooms and slabs are given in Table 3.

Edge reduction of the slabs is usually accomplished on the first stand and if the ingot is too wide then on the first and second stands. Between the last three pairs of stands there are undriven vertical rolls serving only for the removal of the width and for forming the side edges.

In front of each pair of stands and in front of the first and second stands manipulators are installed.

The diameter of the rolls in each stand is 1100 mm, the length of their bodies in the first four stands is 1400 mm and 2300 mm in the rest. Along the edges of the roll bodies of each stand, starting with the fifth, there are two passes.

The distance between the stands in which turning is done was selected from the condition of the arrangement of the longest flat with the necessary reserve. The over-all length of the blooming mill building remained almost the same. The working stands can be equipped with slow-speed screw-down devices with a magnitude of lift of the upper roll of 450-500 mm for the first and second stands and 250-300 mm for the remaining stands. Lifting of the upper roll is required only before starting to roll the next ingot. Manipulators and heavy tables are not necessary.

Rotation of the rolls from the motors is transmitted through a two-step reducer with cylindrical spiral gears (4,6,7,8,9, and 10th stands) and three-step reducers (remaining stands).

A calculation was not made for the capital expenditures for installation of such a stand. We consider that the increase in the weight of the mechanical equipment should,to a considerable degree, be compensated for by its simplicity, by the considerable reduction in the power for the electrical equipment for driving the stands and auxiliaries, by the reduction in the height of the building, by the elimination of the need for two additional stands with horizontal rolls and driven vertical rolls for rolling slabs, by the possibility of complete automation of the rolling process, etc.

The annual output of such a blooming mill with rolling 50% blooms with a cross section of 250 x 250 mm and 50% slabs with a cross section of 100-750 mm, if one takes the duration of the intervals between ingots

as 15 sec, with 7200 working hours per year and the utilization factor of the stand as 0.9, is 3.8 million tons per year even with calculation for rolls of minimum diameter.

Taking into consideration the presence in the assortment of large blooms and slabs and the possibility of reducing the interval between ingots, the productivity of the blooming mill can be raised if necessary to 4.5-5 million tons per year and more.

Thus with capital expenditures approximately equal to the expenditures for installing a two-stand reversing blooming mill, the continuous blooming mill will have unlimited reserves of productivity.

In our opinion the installation of continuous blooming mills is already economically advantageous, starting with an output of 3-3.5 million tons per year. This figure can be determined more exactly after a careful study of the problem by the planning organizations.

* * *

EXPERIENCE ON THE OPERATION AND MODERNIZATION OF THREE-ZONE CONTINUOUS FURNACES

V. N. Medvedev, N. I. Butenko, and V. P. Murovtsev

Alchevsk Metallurgical Works

Three-zone bottom-fired continuous furnaces intended for heating slabs of maximum size $150 \times 800 \times 2000$ mm have been built for the 2250 mill according to the design by "Giprostal' ". A mixture of coke-oven and blast-furnace gas of 2000 kcal heating value at a pressure of 220 mm water is used as fuel.

Air to the furnace is supplied by two fans of 25,000 m^3/hr capacity at a pressure of 360 mm of water. The air is heated up in metallic extended-surface recuperators.

A distinguishing feature of these furnaces is the positioning of the burners in the upper heating zone where they face the burners of the soaking zone (i.e., in the direction of the metal movement).

Square drilling bars are used as bottom beams. The longitudinal beams are set on edge and in the heating zone they are mounted on the supporting double beams of the same cross section with their ends built into the brickwork of the side walls. The transverse supporting beams in the continuous zone are also resting with their middle part on a longitudinal wall.

The furnaces are equipped with instruments for temperature measurement and automatic control.

In the course of the operation of the furnaces some short-comings were noted which resulted in a high specific fuel consumption (up to 190 kg/t low efficiency and poor heating of the metal. The main cause of the short-comings is defects in the design of the funace.

In recent years, the personnel of the Works have modified and improved some parts of the furnaces. In the heating zone additional П-shaped beams have been installed and this has improved the rigidity of the supports and prevented the tubes from bending even if the cooling water supply is interrupted for a short time. The design of the water-cooled supporting beam at the discharge end has been modified because the welded supporting beam used to get fouled rapidly in its lower parts and burned through in 2.5-3 months. The present beam, made of rectangular tubes (drilling bars) has been in operation for over three years. The beam supporting the roof knuckle has been modified in a similar manner.

The rectangular hearth girders of Kh8S steel have been replaced by heavier T-girders made of Kh25H2 cast steel and, as a result, their service life has increased from three to eight months. Even now, however, the replacement of the girders causes additional shutdowns of the furnaces during which time a partial repair to the brickwork, hearth beams, etc., is carried out. In the period between the repairs, the hearth girders are sections, are supported individually with the use of kaolin bricks of a simple shape (Fig. 1). As a result of this the service life of the roof has increased and the amount of work which has to be carried out on the roof during the replacement of the hearth girders has been reduced.

The design of the small shaped joists of the roof knuckle has also been modified with the object of preventing the bricks from falling through the slots made in the lower flange of the joist for convenient anchoring of the bricks. As a result of bricks falling out, the joists burned through and therefore the furnace had to be shut down two or three times a year for the replacement of the brickwork of the roof knuckle. Now the lower flange of the beam is made without slots and the bricks do not fall out.

The covers of the discharge end were suspended on a No. 36 H-beam which warped at high temperature and became useless; its service life has been increased by

cooling it with running water.

Water-cooled welded dampers burned through frequently and their service life did not exceed three months. New cast dampers with refractory linings which have replaced welded ones have been in operation for over two years.

After the improvement of some parts, the service life of some components of the furnace increased but the efficiency of the furnace and the quality of the metal heating remained as before. So a further increase in the output of the mill was hampered.

Observations of the furnace operation and the investigations carried out by the Central Works Laboratory showed that as a result of the pressure produced by the burners of the upper heating zone a positive pressure was established in the heating and soaking zones while the remaining part of the furnace was continually under a negative pressure. The burners in the preheating zone not only did not help to establish a positive pressure but, on the contrary, produced negative pressure at the end of the furnace.

Observation showed that as a result of large temperature differences between the gas and the metal in the beginning of the preheating zone, the heating of the slab proceeds sufficiently quickly, but as soon as the slabs reach the vicinity of the second window the temperature of the slab bottom does not increase because of the high rate of heat transfer to the water-cooled tubes. This happens also at the end of the preheating zone as well as at the beginning of the heating zone.

Therefore the slabs entering the heating zone were at a relatively low temperature (the temperature in the center of the slab was approximately 650°C) and showed a large temperature difference between the top and the bottom of the slabs.

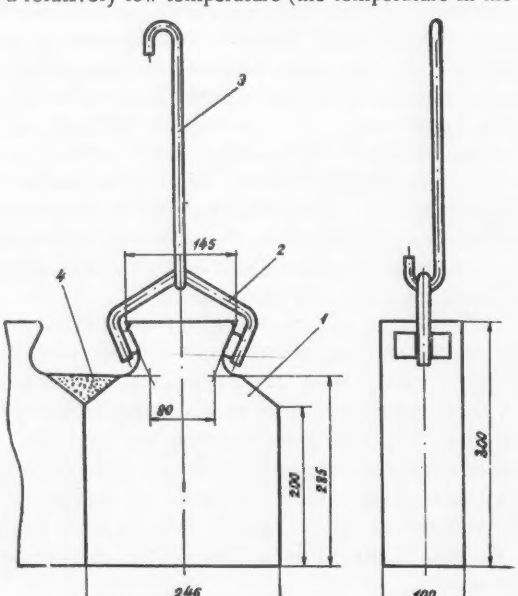


Fig. 1. Individual anchorage of roof brick: 1) suspended brick; 2) bracket; 3) suspension hook; 4) thin mortar mixture.

In the heating zone the gas burned at a high rate and the quantity of heat liberated in the zone was very large; this resulted in a high temperature of the slab surface, but the temperature over the cross section of the slabs still remained nonuniform. The temperature of the slabs did not become uniform even in the soaking zone because, before they were discharged, the slabs rested on the water-cooled threshold and the hearth in that part of the furnace was also cool due to the leakage of cold air into the furnace. Therefore, the temperature difference over the cross section of the slab delivered for rolling was 250-300°C.

As a result of the high temperature of the surface of the slabs in the soaking zone, there was a rapid fouling of the furnace bottom and it had to be cleaned at least every week, which involved a great deal of work.

A high fuel consumption in the furnaces was due to a poor functioning of the recuperators and also to high heat losses with the cooling water. Measurements showed that the heat losses with the cooling water constituted 25-27% because of the large number of water-cooled tubes in the furnace and the fact that they were not insulated in the heating and preheating zones. A large quantity of heat was lost with gases forced out through the sight window.

It was decided to carry out a radical modernization to improve the operation of the furnaces.

The modernization project was developed by the Design Section at the Works, on the basis of the suggestions of the technical personnel of the shop and it was carried out during the major overhaul of the mill in 1957.

During the modernization, the burners of the upper heating zone were spread through 180° deg and the direction of the burners of the lower heating zone was changed by setting them at an angle of 8 deg to the slabs being heated (Fig. 2). The roof of the preheating zone was made straight and sloping toward the charging side. To reduce the gases coming out of the charging door at the charging end of the furnace, the roof over that end of the furnace was lifted so that a chamber, which reduced the velocity of the gases, was formed. The burners of the preheating zone were removed. A part of the Π-shaped supporting beams in the heating zone was removed and the number of transverse beams of the continuous zone was halved, and, in addition, all water-cooled fenders were eliminated. As was pointed out earlier, because of the gaps between the longitudinal hearth beams, the lower side of the slabs was cooled and they were displaced laterally. Therefore, the longitudinal beams were made continuous by butt welding. In the preheating zone they were laid on strong steel supports mounted on the longitudinal brick walls. The sides of the beams were covered with refractory bricks. In this way the problem of the insulation of the

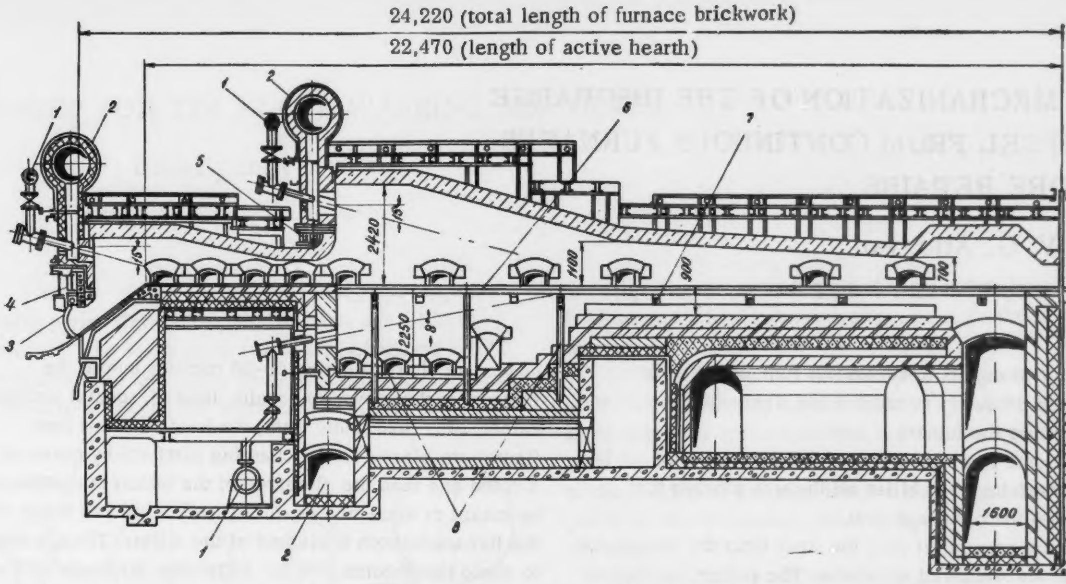


Fig. 2. Three-zone continuous furnace of the 2250 mill after the modernization: 1) gas header; 2) hot air header; 3) water-cooled beam at the discharge end; 4) suspension beam for doors; 5) supporting beam for hot air header; 6) longitudinal beam skids; 7) transverse supporting beams; 8) Π-shaped supporting beams.

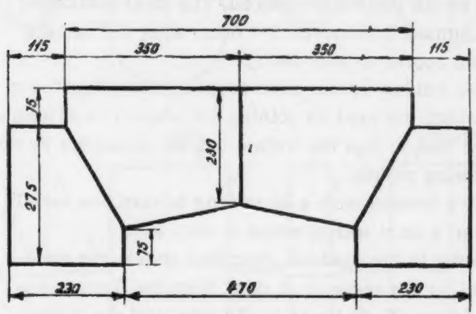


Fig. 3. The arch of the side window laid of blocks: 1) block; 2) abutment brick.

skids in the preheating zone was solved and they were prevented from sagging so that the slabs could be kept in regular rows during their journey along the furnace.

The vertical part of the supporting beams in the heating zone was insulated by chrome-magnesite brick interlaid with sheets of black plate. At a high temperature the bricks and the black plate form a solid continuous mass which can withstand even heavy impacts. The horizontal part of the longitudinal and supporting beams in the heating zone was insulated with reinforced chromite blocks but this insulation does not last for more than three months.

The sight windows were assembled from two blocks (Fig. 3) for a better strength of the arch.

All this considerably improved the heating regime in the furnaces. The reduction of the number of water-cooled beams resulted in a decrease in the heat losses to the cooling water by at least 35%. The heating regime of the steel in each zone improved. The steel which leaves the preheating zone is well heated through its whole thickness when it enters the heating zone; owing to the fact that the burners of the lower heating zone are set up at an angle to the surface of the heated steel, the flame is kept tight against the metal surface and thus the rate of heat transfer is increased and the process of heating from below becomes more effective. An improvement in the distribution of the pressure in the working space of the furnace helps in reducing the forcing-out of the flame, reduces the heat losses through the sight windows and the discharge door, and increases the service life of the furnace equipment. As a result of the reduction in the leakage of cold air into the soaking zone and owing to the improvement in the quality of the heating of the steel there is almost no fouling of the soaking zone bottom.

After the modernization of the furnaces and repairs to the recuperators it became possible to introduce automatic control of furnace operation.

The improvement of the furnaces made it possible to reduce the specific fuel consumption to 153 kg/t; the productivity of the furnaces increased by at least 20% and, at the same time, the quality of the heating of the metal improved considerably.

* * *

THE MECHANIZATION OF THE DISCHARGE OF STEEL FROM CONTINUOUS FURNACES BEFORE REPAIRS

V. G. Attaryan

Tube Billet Mill at the Zakavkaz Metallurgical Works

Prior to repairs to continuous furnaces at rolling mills it is necessary to remove the steel which has been charged into the furnace.

At some steel works this operation is carried out by hand by turning each billet all the way through the furnace to the discharge door.

At our tube billet mill the steel from the continuous furnaces was removed as follows: The pusher was moved in; several billets of a total width equal to the length of the movement path of the pusher were held with a chain and the ends of the chain were attached to a special hook welded to the pusher rods. The pusher was then set in reverse motion and the billets were pulled out through the charging door, placed on the rolling tables and taken away by the crane. This procedure was repeated until all the billets were removed from the furnace.

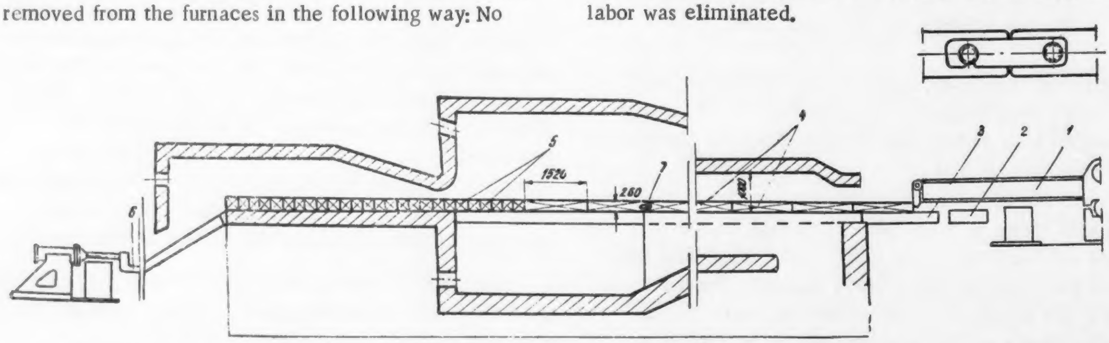
The use of this method involved a preliminary cooling of the furnace together with its charge (for at least 24 hours) in order to enable the workers to enter the furnace and put the chains around the billets; but even then the work in the furnace was very arduous.

On the proposal of the furnace foreman at the tube billet mill, G. N. Feshchenko, the steel billets are now removed from the furnaces in the following way: No billets are charged during 20-30 minutes before the furnace is shut down for repairs; instead, special welded frames, 260×1520×3000 mm, made of channels (see figure) are placed on the charging platform by means of a crane and then the discharge of the billets is continued by means of these frames. In this way up to two thirds of the furnace bottom is cleared of the billets. Then, in order to avoid the distortion of the frames, the discharge of the billets is discontinued and the furnace is shut down for repairs. The furnace, cleared of two-thirds of the billets, cools down more quickly and, therefore, the shutdown period of the furnace is reduced. The steel remaining in the furnace is removed 2-3 hours after the furnace has been cooled to 500-600°C.

The welded frames have coupling bars (figure, top right) which are used for joining the whole row of frames into one unit so that the frames can be pulled out by the withdrawing pusher.

For a furnace with a 26 m long bottom one uses 18 frames of a total weight equal to 20.5 tons.

Owing to the method, described above, the time required for the removal of steel from the furnace was reduced from 20-24 hrs to 60-70 min and the arduous labor was eliminated.



A continuous furnace with hot billets being discharged with the use of special frames: 1) pusher; 2) feed roller tables; 3) charging platform; 4) frames for pushing out the billets; 5) billets; 6) discharge roller tables; 7) frame coupling.

* * *

GUIDES FOR TIN PLATE WASHING MACHINES

V. P. Emel'yanov

Magnitogorsk Metallurgical Combine

After the rolling process the black plate strips are usually passed through degreasing baths and washing machines. The black plate is welded beforehand into a continuous strip and passes continously through the baths. If the strip breaks in the degreasing bath the whole production is interrupted for a considerable time because it is necessary to catch the broken end and take it through the washing machine by hand.

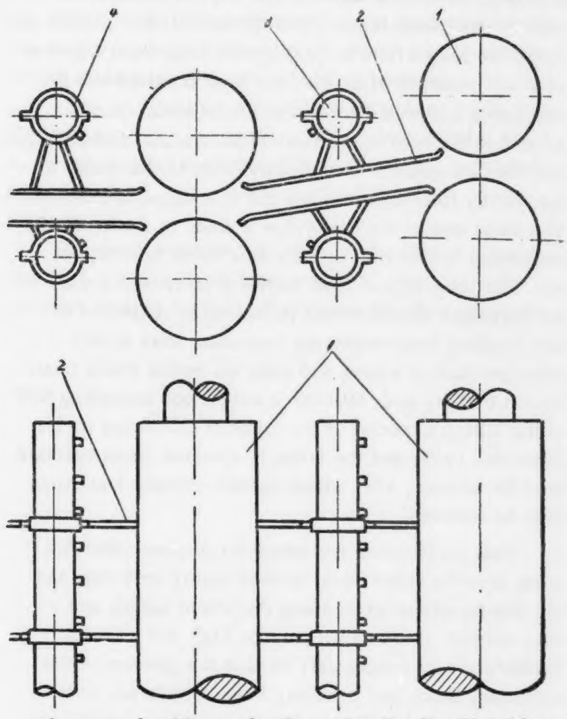


Diagram of guide mechanism: 1) Rolls; 2) roll guides; 3) header; 4) yoke.

In the washing machine the strips are washed with hot water. If the strip breaks, the water must be let out since otherwise it is impossible to fix the end of the strip between the rolls. To stop the supply of hot water to the headers of the machine one has to shut down the pump for 30-40 minutes and that again extends the stoppage.

At the tin plate shop of the Magnitogorsk Metallurgical Combine, on the proposal of N. A. Kadoshnikov guides (see figure) were installed between the rolls with the object of preventing broken ends of the strip from falling to the bottom of the bath and in order to facilitate the refixing of broken ends between the rolls of the washing machine.

The guides consist of separate steel rods of 7-8 mm diameter with bent ends. Because of the small diameter of the rods there is an easy access of hot water to the strip even in the places where the rods are accommodated. Jets of hot water coming from the nozzles in the header splash all over the strip and wash it, including the places covered by the guides. Each guide is connected by means of supports to a yoke secured around the header. Between each pair of rolls there are two guides above and two guides below the strip. If the strip breaks, it is retained on these guides and can easily be reentered between the rolls.

As a result of the introduction of this device stoppages of the washing machines due to breakages of the strip have been reduced.

* * *

INCREASE OF THE LONGEVITY OF STEEL CABLES

N. N. Grishaev

"Zaporozhstal' " Plant

Ordinary steel cables are insufficiently resistant to wear and are not dependable in service on hoists, leading to frequent interruptions in production and a large expenditure of cables. (It has been calculated that 0.055 kg of open hearth steel, 0.009 kg of slabs, 0.008 kg of hot-rolled sheet, 0.024 kg of cold-rolled sheet, and

0.07 kg of steel cable are consumed during the manufacture of each ton of finished cast iron). The causes of this are as follows:

1. Immediately after the introduction of cables into service 7 to 8 (and in some cases 10 to 12) wires per foot of twist are ruptured, which represents 50% of the standard for rejection of steel cable. The single wires break because they are subjected to unequal tension during weaving into strands.

2. Separate strands begin to bulge during the first days of service; a cable with such a defect is rejected, the piece of cable containing the bulge is cut out, and the rest of the cable is used for less critical work. The bulges are also the result of unequal stress of strands during weaving.

3. Abrasion of the wires is first very slow—up to 15% of the total wear—but later increases sharply. When the wires are 35-40% abraded they break. This is caused by incorrect normalizing conditions.

At the "Zaporozhstal'" Plant it was decided to elucidate the cause of abrasion of wires. Two pieces of cable of the same type, construction, and dimensions were used; one piece was provided by the Khartsyz Plant and the other was cut from an imported cable.

Both cables had the same chemical composition but different mechanical characteristics. The second (imported) cable had been in use for 11-12 months on the hoists of blast furnaces—its longevity being 2.5 times that of the cable from the Khartsyz Plant. The wires of the imported cable were compressed equally through the whole cross section, abrasion of the wires was even, and the wires broke only after they were 50-60% worn.

The investigation showed that abrasion and rupture occur mainly in the areas in contact with stationary blocks and with blocks of suspended hooks because in these areas steel rubs against steel (cast iron blocks not being used in heavy duty cranes).

To decrease abrasion it is necessary to coat the sheave grooves with a 5-6 mm layer of bronze; the life of the cable increases up to one year. To reduce breaking of the wires the stationary block must be designed so that the diameter is not decreased by 40% as recommended in the GNK ruling (article 7).

At the present time cables are lubricated with cable grease and waste oil. This type of lubrication is quite satisfactory in shops where the atmosphere does not contain dust in suspension, but in very dusty shops (blast furnace, open hearth, and stamping shop) the lubricant becomes mixed with dust, forming dirt, and becoming a basic source of abrasive wear. In such shops it is necessary to lubricate the grooves on the sheaves, and not the cables.

The length of cables is often measured by laying them on dusty floors. They become dirty, thus accelerating wear. To prevent this, the cable drum must be brought from the warehouse to the shop and the cable suspended without being laid on the floor. This new method does not dirty the cable and reduces mounting time from 4-8 hours to 45 minutes. It is as follows: The hook is lowered to the floor and the end of the new cable is placed around the sheave; also the old cable is cut with an acetylene torch. Then the ends of the old and new cable are joined (end to end) by stitching them together with soft weaving wire; then the hook is raised and the new cable is strung through the whole block system. After this the ends of the two cables are disconnected and the new cable is temporarily fixed to the frame of the trolley (bridge) while the old is lowered to the floor. The other end of the new cable is fixed to the block, the suspension is checked, and the new cable is ready for use. The longevity of steel cables is increased if they are burnished and stretched before being subjected to full working loads—rupturing decreases down to 2-3 wires per foot of weave and abrasion begins much later. This stretching is as follows: A weight not exceeding 50% of the lifting capacity of the crane is connected to the suspended cable and the crane is operated under medium load for an hour, after which normal working conditions may be resumed.

Thus, to increase the longevity of steel cables the wires must be under equal tension during weaving, and the density of the wires along the whole length and cross section of the cable must be high and uniform. Furthermore, it is necessary to coat the grooves of the stationary block and traveling block with brass, and lubricate the grooves periodically.

Plants using steel cables must burnish and stretch them before putting them into operation, and also suspend them by the new method described.

When all these measures are taken the longevity of steel cables is increased 2 to 2.5 times.

* * *

I. S. Trishevskii, Guides for Rolling Mills (Metallurgizdat, Kharkov, 1957. 283 pages).

Various designs of guides and their operating conditions on merchant mills are examined in the book; methods for installing the guides are given and the basic parameters of the guides and edging rolls are determined. A method is given for designing, manufacturing, and adjusting the guides, thus assuring a true setting installation, and increase in the service life of the rolls.

The causes for wear and maintenance are cited as well as the materials for fabricating the guides.

The book is intended for engineers and technicians, but is also of use to workers in the merchant-mill shops and to students at institutes of higher learning.

The Efficiency of Various Methods of Heating the Risers (Translated from English by A. P. Perov under the editorship of Doctor of Technical Sciences, G. N. Oyks. Metallurgizdat, Moscow, 1959. 57 pages.)

This book contains two works of the British metallurgists J. Fenton, G. S. Marr, and V. G. Glesher, who examine various methods of reducing the riser of ingots. Many of these methods, e.g., heating the metal by means of an electric arc, covering the surface of the metal with lunkerite, etc., are known to us and have been tested under factory conditions. In spite of this it is desireable that Soviet metallurgists become familiar with the works of their British colleagues. Heat currents in the riser of ingots are analyzed in the article by J. Fenton. Experiments were conducted on 230 x 230 ingots weighing 250 kg. It was established during the investigations that the greater part of the heat (about 75%) released by the riser during solidification is absorbed by the refractory material of the adapter. Further an adapter making it possible to reduce the loss of metal by 2.5% is described; this was developed by the British Ferrous Metallurgy Research Association.

In the work of G. S. Marr, J. Fenton, and V. G. Glesher, recommendations based on careful laboratory investigations are given which are of great interest to Soviet steel makers.

The book is provided with an introduction by Doctor of Technical Sciences G. H. Oyks in which a critical analysis of both works is given.

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A CAROUSEL-TYPE FURNACE IN A PIPE-ROLLING MILL

A. Kopachek

Chairman of the Central Committee of the Workers' Trade Union of the Metallurgical Industry and Mines of the Czechoslovak Republic.

Clement Gottwald "Novaya Gut" Combine at Kunchitsakh

Since the beginning of 1959, in the pipe-rolling mill of the combine, a new carousel-type furnace has been used to heat round billets for the production of seamless pipes. The funace was planned by the Gutnyi Proekt Trshinets and built by the Klatov Engineering Plants. Its planned productivity is 40 tons/hr; the furnace can take round billets of diameter 100-250 mm, of maximum length 3,800 mm.

The furnace is fired by a mixed gas of calorific value 1600 kcal, the gas consumption being 16,000 m^3/hr. The average diameter of the furnace is 18 m, external diameter 24.5 m, width of hearth 4 m, the lining is of firebrick, weight of the hearth 400 tons. The average time forheating the billets is 2 hours.

The carousel-type furnace replaces two previous rerolling furnaces which were serviced during the day by 75 workers. Their work was very difficult since the temperatures were rather high and they rerolled the billet by hand using iron bars. There were frequent accidents due to stumbling over the bars, burning the hands, etc.

The operation of the new carousel-type furnace requires about 15 men per day. The furnace is fully automatic. The billets pass along the roller table to a loading machine which is controlled by an operator from a control cabin. The heated billets are also removed by a machine which is similar in design to the charging machine. During the whole time the equipment has operated at the furnace there has been no case of breakage in the equipment and no serious breakdown.

The use of the carousel-type furnace has increased the productivity of the section by 80%, savings in wages being about 1 million rubles per year and the total saving about 2 million rubles per year.

* * *

EQUIPMENT FOR INSULATING PIPES BY SPRAYING IN AN ELECTROSTATIC FIELD

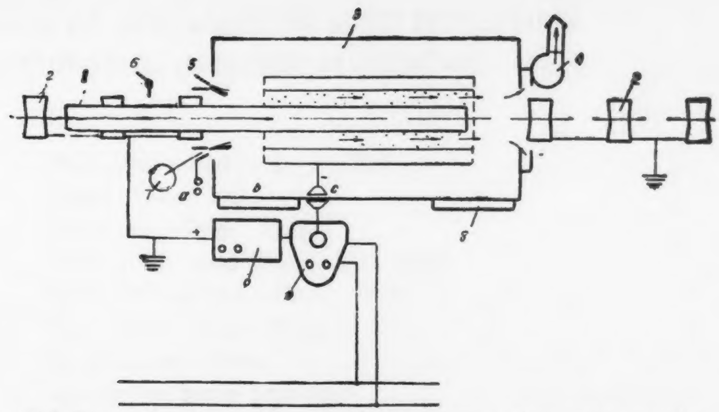
A. Kopachek

Czechoslovakia

At the present time an important problem in pipe-rolling mills is the method for insulating the pipes, since during insulation a considerable amount of insulation material is still expended (oil, special asphalt varnishes). The pipes are treated with oil manually, which causes a considerable amount of eczema in the workers, and, furthermore, does not guarantee uniform application of the insulation materials on the surface.

The Chief Planning Engineer of the "Novaya Gut' " Combine, Jan Burda, proposed the insulation of pipes by spraying in an electrostatic field. For this purpose, an experimental unit was developed, tests on which have been completely successful.

The unit works on the same principle as the electrostatic cleaning of blast-furnace gas. The insulating material is sprayed by means of special automatic pulverizing guns into the space around a horizontal electrode cylinder, formed by fine wires. The cylinder is connected to the negative pole of a dc high-voltage generator and in the middle of the electrode there is the moving pipe which is to be insulated. The pipe is connected to the grounded positive pole of the same generator. In this space an electrostatic field is set up at more than 100,000 v. The very finely atomized drops of the insulation material in the form of a fog are given an electric charge and are attracted toward the grounded pipe.



Unit for coating pipes with an insulating layer: 1) Pipe; 2) roller table; 3) spray chamber; 4) exhaust fan; 5) gun; 6) photocell; 7) reservoir for insulation material; 8) control device; 9) control panel; 10) high-voltage generator; a) solenoid valves; b) electronic panel; c) high-voltage conductor.

The main difference between the usual method for spraying and spraying in an electrostatic field is that during ordinary spraying the spray gun should be directed at the object. When the material was reflected from the object or fell near it, there were large losses. In electrostatic spraying the gun is not pointed directly at the object but the stream is directed along the object in the space between the object and the electrode. The drops are attracted toward the object from all sides; there is no splashing from the material nor does the material miss the object. The losses are therefore very small.

This method also ensures a very even coating of the material.

The equipment (diagram) consists of a spray chamber through which the pipe intended for coating passes along a roller table. The speed of the roller table is controlled over wide limits from a control panel. Special guns are used with very fine atomizers. The guns work automatically on compressed air.

The control device reduces the working pressure of the compressed air to the required value and at the same time removes unwanted moisture. The low-pressure air is led to the reservoir for the insulation material and into the spray guns through solenoid valves which are controlled by a photorelay in an electronic control panel receiving an impulse from a photocell.

As soon as the front end of the pipe passes the photocell, a photorelay is brought into action which starts the passage of air in the solenoid valves and the guns begin to spray. In the electronic panel there is a retarding relay which acts in such a way that the guns continue to spray for a given number of seconds after the back end of the tube passes the photoelement so that the whole of the tube is treated. If there should be an excess of the finely atomized insulation material, it is drawn off by an exhaust fan and condensed.

The high-voltage direct current, reaching 150,000 v is provided by a high-voltage generator. The electrode cylinder is connected by a cable passing through the wall of the chamber to the high-tension conductor with the negative pole of the generator; the pipe is connected through the roller table with the grounded pole.

The equipment has the essential signals and blocking arrangements to make the operation perfectly safe.

This unit can deal with pipes of 114-245 mm diameter.

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ERRATA

Volume 7, July, 1959, Page 288

Left column, second paragraph, first sentence reads:

For each steel grade the mean and the most frequent temperatures, the mean and the mode and the standard deviation were determined statistically (Table 1).

It should read:

For each steel grade the mean and the most frequent temperatures, and the standard deviation were determined statistically (Table 1).

MISSING PAGES ARE INDEX PAGES
WHICH HAVE BEEN PHOTOGRAPHED
AT THE BEGINNING OF THE VOLUME(S)

SIGNIFICANCE OF ABBREVIATIONS MOST FREQUENTLY ENCOUNTERED IN SOVIET PERIODICALS

FIAN	Phys. Inst. Acad. Sci. USSR.
GDI	Water Power Inst.
GITI	State Sci.-Tech. Press
GITTL	State Tech. and Theor. Lit. Press
GONTI	State United Sci.-Tech. Press
Gosénergoizdat	State Power Engr. Press
Goskhimizdat	State Chem. Press
GOST	All-Union State Standard
GTTI	State Tech. and Theor. Lit. Press
IL	Foreign Lit. Press
ISN (Izd. Sov. Nauk)	Soviet Science Press
Izd. AN SSSR	Acad. Sci. USSR Press
Izd. MGU	Moscow State Univ. Press
LÉIIZhT	Leningrad Power Inst. of Railroad Engineering
LÉT	Leningrad Elec. Engr. School
LÉTI	Leningrad Electrotechnical Inst.
LÉTIIZhT	Leningrad Electrical Engineering Research Inst. of Railroad Engr.
Mashgiz	State Sci.-Tech. Press for Machine Construction Lit.
MÉP	Ministry of Electrotechnical Industry
MÉS	Ministry of Electrical Power Plants
MÉSÉP	Ministry of Electrical Power Plants and the Electrical Industry
MGU	Moscow State Univ.
MKhTi	Moscow Inst. Chem. Tech.
MOPI	Moscow Regional Pedagogical Inst.
MSP	Ministry of Industrial Construction
NIIZVUKSZAPIOI	Scientific Research Inst. of Sound Recording
NIKFI	Sci. Inst. of Modern Motion Picture Photography
ONTI	United Sci.-Tech. Press
OTI	Division of Technical Information
OTN	Div. Tech. Sci.
Stroiizdat	Construction Press
TOÉ	Association of Power Engineers
TsKTI	Central Research Inst. for Boilers and Turbines
TsNIÉL	Central Scientific Research Elec. Engr. Lab.
TsNIÉL-MÉS	Central Scientific Research Elec. Engr. Lab.-Ministry of Electric Power Plants
TsVTI	Central Office of Economic Information
UF	Ural Branch
VIÉSKh	All-Union Inst. of Rural Elec. Power Stations
VNIIM	All-Union Scientific Research Inst. of Meteorology
VNIIZhDT	All-Union Scientific Research Inst. of Railroad Engineering
VTI	All-Union Thermotech. Inst.
VZÉI	All-Union Power Correspondence Inst.

Note: Abbreviations not on this list and not explained in the translation have been transliterated, no further information about their significance being available to us - Publisher.